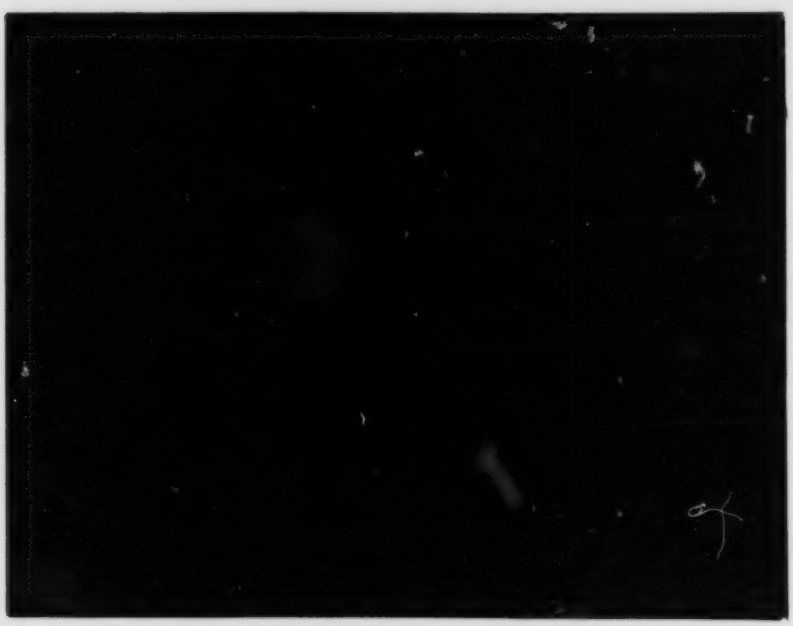
MINERALS

ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT



Canada

Manitoba

MG-13097A

ECONOMIC ASSESSMENT AND PRE-FEASIBILITY STUDIES

ON THE BIRD RIVER CHROMITE DEPOSITS

as part of the

MANITOBA MINERAL DEVELOPMENT AGREEMENT

D.S.S. Contract Serial No. 23440-6-9147/01-SQ

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FINAL REPORT

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MICROMEDIA

Hatch Associates Ltd. Project No. 9075

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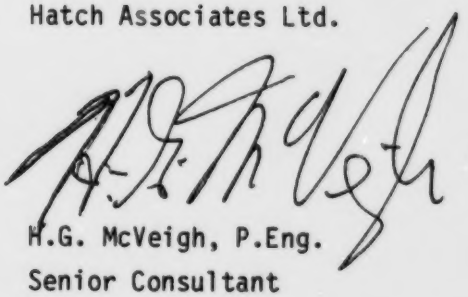
Hatch Associates Ltd.



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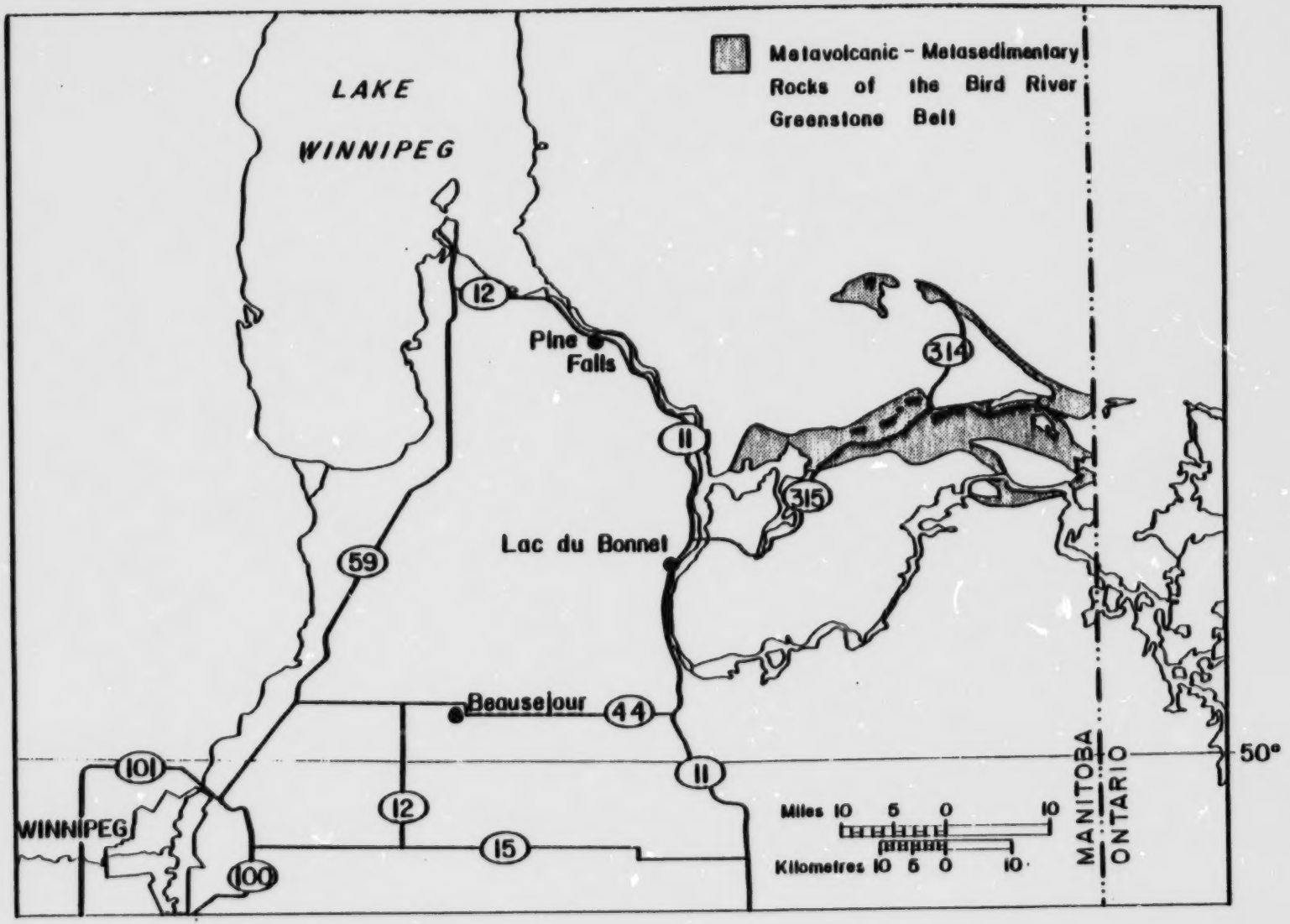

Index map, southeastern Manitoba (Bird River Sill shown in solid black)

THE BIRD RIVER CHROMITE DEPOSITS

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1.0 INTRODUCTION

As part of the Canada/Manitoba Development Agreement, Hatch Associates were commissioned by Energy Mines and Resources, Canada, to prepare a pre-feasibility evaluation of the possible future economic development of the Bird River Chromite Deposits. This is identified as Phase III of a presently proposed four-phase program.

As a basis for the Phase III work the Phase II report by Lakefield Research on the beneficiation of a 70 tonne bulk sample of material taken from the "Chrome Deposit" was used.

In addition, information gathered from previous geological reports (see Appendix 1), and also supplied from a meeting held with Federal and Manitoba representatives in Ottawa on February 9, 1987 have been taken into account.

1.0 INTRODUCTION

Dans le cadre de l'entente Canada/Manitoba, le Ministère de l'Energie, des Mines et des Ressources du Canada a mandaté Hatch & Associés pour préparer une évaluation de la faisabilité économique pour le développement éventuel des dépôts de chromate de la rivière Bird. Ceci est identifié comme étant la phase II du présent programme tel que proposé en quatre phases.

Le rapport de la phase II, préparé par Lakefield Research et portant sur le traitement d'un échantillon composite de 70 tonnes de matériel provenant du dépôt de chrome, a été utilisé comme base de travail pour la phase III.

De plus, des informations colligées à partir de rapports géologiques antérieurs (voir l'annexe 1) de même que des informations obtenues lors d'une rencontre entre des représentants du fédéral et du Manitoba (Ottawa, le 9 février 1987) ont été utilisées pour cette évaluation.

2.0 CHROMIUM AND THE NORTH AMERICAN MARKET

Prior to studying the following recommendations and preliminary technical assessments of this Phase III report, we consider it may be of value to the reader to be appraised of the present supply and demands of the North American market place.

The present and traditional supply of chromium, in the form of "ferro-chromium" to the steel industry is known as "charge chrome". The minimum chromium content in these ferrochromium alloys is 50 percent, and they are typically produced from chromite concentrates analysing 45 percent chromium oxide with a chrome to iron ratio of 1.6:1.

Ninety-five percent of North America's ferrochromium demand is imported. It is shown (see Table 1) that stainless steel and heat resistance steels produced from these imports in the United States during 1986 approached 1,700,000 net tons and consumed approximately 151,000 net tons of chromium. This equates to in excess of 300,000 net tons of ferrochrome (see Table 2, which delineates the sources). (Note: Atlas Steel, Canada, imports their requirements from Rio Tinto's operations in Zimbabwe, which represent an additional 8 to 10 percent to the amounts shown).

The manufacturing capacity in the United States for stainless steel sheet and strip is dominated by Allegheny Ludlum, Armco and J&L Specialty, all of whom are located in the Pittsburgh, Pennsylvania area. Their production consists of two distinct and separate operations: firstly, the melting and alloying of the steel in the steel mill, to produce an acceptable alloyed product in the form of an ingot, slab or billet - "semi-finished steel"; and secondly, the reheating and then rolling of the steel into a form suitable for use by the manufacturing industries. Traditionally, the rolling capacity in a major integrated operations is substantially in excess of its melting (steel producing) capacity.

Chromium-nickel stainless (the designated 200/300 series steels) account for 75 percent and straight-chrome stainless (the 400 series steels) for 25 percent of the market. Sheet and strip steel products of both series of grades amount to 60 percent of the total stainless market. Presently, 20 percent of this market is being penetrated by imports, primarily from Europe and Japan.

2.0 LE CHROME ET LE MARCHE NORD-AMERICAIN

Avant d'étudier les recommandations et les évaluations techniques préliminaires de ce rapport sur la phase III, nous considérons qu'il peut être utile pour le lecteur d'être informé sur le marché nord-américain actuel, en ce qui a trait à l'offre et la demande.

Traditionnellement et encore aujourd'hui, l'approvisionnement en chrome, sous forme de ferro-chrome, dans l'industrie de l'acier est connu sous l'appellation "chrome de charge". La teneur minimale en chrome de ces alliages ferro-chrome est de 50% . Ils sont typiquement produits à partir de concentrés de chromite contenant 45% d'oxyde de chrome, ce qui donne un rapport chrome: fer de 1.6:1.0.

Quatre-vingt quinze pourcent de la demande nord-américaine en ferro-chrome est satisfaite par l'importation. On constate (voir tableau 1) que les aciers inoxydables et les aciers à haute résistance thermique produits aux Etats-Unis à partir de ces importations totalisaient 1 700 000 tonnes en 1986 et engendraient une consommation en chrome d'environ 151 000 tonnes. Il en résulte un excès de 300 000 tonnes de ferro-chrome (voir au tableau 2 pour la description des sources). (Note: Atlas Steel Canada s'alimente à partir des installations de Rio Tinto au Zimbabwe, ce qui représente un montant additionnel de 8 à 10% par rapport au montant indiqué.)

La capacité manufacturière des Etats-Unis pour les aciers inoxydables en tôles et en feuillards est dominée par Allegheny Ludlum, Armco, et J & L Specialty, compagnies qui sont toutes localisées dans la région de Pittsburgh en Pennsylvanie. Leur production consiste en deux opérations distinctes et séparées: premièrement, la fonte et l'alliage de l'acier dans l'aciérie pour obtenir un produit acceptable sous forme de lingot, brame ou billette - "acier semi-fini"; deuxièmement, le rechauffage et le laminage de l'acier en une forme plus adaptée pour une utilisation par les industries manufacturières. Traditionnellement, dans un complexe important avec des opérations intégrées, la capacité de laminage excède fortement la capacité de fusion.

Les aciers inoxydables chrome-nickel (identifiés dans la série 200/300 des aciers) accaparent 75% du marché et les aciers inoxydables ordinaires au chrome occupent 25% du marché. Les produits en acier, tôles et brames, des deux catégories de teneur totalisent 60% du marché total des aciers inoxydables. Actuellement, 20% de ce marché est pénétré par les importations, principalement d'Europe et du Japon.

TABLE 1

UNITED STATES STEEL INDUSTRY DATA

	Total Raw Steel Production (Net Tons)	Stainless & Heat Resisting Steel Production (Net Tons)	Primary Chromium Units Consumed per net ton S&HR Steel Production (pounds)	Total Primary Chromium Consumed in the S&HR Steel Sector (Net Tons)
1980	111,835,000	1,701,000	206	175,200
1981	120,828,000	1,743,000	201	175,200
1982	74,577,000	1,235,000	180	111,200
1983	84,615,000	1,750,000	210	183,800
1984	92,527,000	1,772,000	191	169,200
1985	87,346,000	1,683,000	191	160,785
1986	80,470,000(P)	1,686,000(P)	179	150,890(P)

Data sources: AISI, USBM.

NOTE: 160,785 NT primary chromium @ 54% average Cr $\cong$ 300,000 NTPA ferrochrome.

P: Preliminary (using December 1986 data)

Source: U.S. Bureau of the Census.

Table 2 - U.S. imports for consumption of ferrochromium in November 1986, by country
(Short tons)

Country	Low-carbon ferrochromium			High-carbon ferrochromium		
	Gross weight	Chromium content	Year to date (gross weight)	Gross weight	Chromium content	Year to date (gross weight)
Brazil	--	--	--	--	--	8,047
Canada	--	--	--	--	--	107
China	--	--	--	--	--	1,102
Finland	--	--	--	--	--	11,028
Denmark 1/	19	14	19	--	--	--
Germany, Fed. Rep. of	866	615	7,157	--	--	--
Greece	--	--	--	--	--	4,409
Italy	19	14	370	--	--	--
Japan	--	--	--	43	29	271
Norway	--	--	57	--	--	--
South Africa, Rep. of	--	--	14,205	15,337	8,004	200,928
Sweden	307	222	4,735	--	--	--
Turkey	--	--	4,960	5,732	3,611	43,970
Yugoslavia	--	--	--	1,543	920	24,645
Zimbabwe	--	--	6,033	--	--	37,334
Total 2/	1,211	865	37,537	22,655	12,564	331,841

1/ Being questioned.
2/ Data may not add to totals shown because of independent rounding.

Source: U.S. Bureau of the Census.

3.0 SUMMARY AND RECOMMENDATIONS

3.1 Summary

The following sections of this report address themselves to general comments of the geology of the area (which has thoroughly been reported by others - see Appendix) and specific comments relating to the economic geology of the "Chrome Deposit" on which this report is based.

This is followed by a preliminary evaluation of possible mining alternatives, and appraisal of potential mineable "ore" reserves, scale of operation and costs.

Section 7 devotes itself to the concentrator. Capital and operating costs have been developed based on the Phase II Lakefield report, as interpreted to provide a maximum recovery of contained chromium at minimum cost.

In Section 8, the concentrates are converted to a marketable product in the form of stainless steel slabs. Flowsheets, capital and operating costs have been developed for a scale of operation which is considered feasible for penetration into the marketplace. These costs are developed on the basis of a fully integrated operation, mine through smelter, being located in the Bird River area.

The following section includes some sensitivity analyses and comments relative to scale of operations, profits and costs.

The Appendix contains the applicable preliminary calculations for the foregoing sections and references.

3.2 Recommendations

3.2.1 A preliminary assessment of the marketplace indicates that the production of 200,000 tons per year of stainless steel slabs (semi-finished steel) is viable.

3.2.2 Based on the Phase II report, this will require the production of 192,000 tons per year of 30.0% Cr_2O^3 concentrates from a mill operating on approximately 3,700 tons per day of mined material.

3.2.3 The mined "ore" requirement is a nominal 1,350,000 tons per year.

3.2.4 A market study should be conducted, or included as part of the proposed Phase IV work. This study should be extended to explore the market for "hot band" and possibly "cold rolled" product as these would represent substantial "added value" to "semi-finished" production.

3.2.5 As stated in 3.2.2, the mill capacity is recommended to be approximately 3,700 tons per day to provide the 30% Cr_2O_3 concentrate. Concentrate production will be 14% of the initial feed to the mill. We recommend that the concentrator's design be based on heavy media separation followed by gravity concentration circuits to provide a suitable concentrate with maximum efficiency and economy.

3.2.6 The mining method for the "Chrome Deposit" should be "open-pit", operating sequentially along its strike length. The primary cut depth is recommended to be 50 metres. The secondary cut depth is a further 50 metres. The pit slope should be 55°.

3.2.7 The Phase IV program should be initiated to refine and develop maximum economy of process and infrastructure; various sensitivity analyses as they effect cost of production; and the market.

3.2.8 It is understood and recommended that prior to production, the following work on the "Chrome" property should be carried out:

(a) additional surface trenching for sampling of the foot wall deposit.
(b) separate samples, rather than a bulk sample, taken in the main zone, to determine location of minerals and grade of each zone component.
(c) a drilling and core-sampling program on a grid to delineate the tectonics of the property.
(d) a pilot plant on site for optimization of mill process and concentrate recovery.

3.0 SOMMAIRE ET RECOMMANDATIONS

3.1 Sommaire

Les sections suivantes de ce rapport réfèrent aux observations générales sur la géologie de la région (voir en annexe les études sur le sujet) et aux observations spécifiques reliées à l'aspect économique de la géologie du dépôt de chrome, aspect sur lequel est basé ce rapport.

Ensuite, une évaluation préliminaire des possibilités d'exploitation minière et une estimation des réserves potentielles de minerai exploitable sont présentées, de même que les paramètres et les coûts d'opération.

La section 7 est consacrée au concentrateur. Les coûts en capital et les coûts d'opération ont été développés en se basant sur le rapport de Lakefield sur la phase II, de façon à obtenir une récupération maximale du chrome à un coût minimal.

Dans la section 8, le minerai concentré est converti en un produit vendable sous forme de brames en acier inoxydable. Les diagrammes de procédé, les coûts en capital et les coûts d'opération ont été développés considérant une opération à une échelle permettant la pénétration des marchés. Ces coûts ont été développés, assumant une intégration complète des opérations, toutes les installations (depuis la mine jusqu'à la fonderie) étant localisées dans la région de la rivière Bird.

La section suivante présente des analyses de sensibilité à différents facteurs et des observations relatives à l'échelle des opérations, aux profits et aux coûts.

L'annexe contient les calculs préliminaires pour les sections antérieures ainsi que les références.

3.2 Recommandations

3.2.1 Une évaluation préliminaire du marché indique que la production de 200 000 tonnes par année de brames d'acier inoxydable (produits semi-finis) est viable économiquement.

3.2.2 En se basant sur le rapport de la phase II, cela nécessitera une production de 192 000 tonnes par année de concentré à 30.0% de Cr_2O_3, à partir d'une usine opérant avec environ 3 700 tonnes par jour de matériel extrait de la mine.

3.2.3 La quantité minimale de minerai à extraire pour la viabilité économique est de 1 350 000 tonnes par année.

3.2.4 Une étude de marché doit être effectuée; elle peut constituer une partie du travail proposé dans la phase IV. Cette étude doit être prolongée pour explorer le marché pour le laminage à chaud ("hot bands") et possiblement pour les produits laminés à froid , ceux-ci pouvant représenter une valeur additionnelle importante pour les produits semi-finis.

3.2.5 Tel que mentionné à la section 3.2.2, la capacité recommandée de l'usine est de 3 700 tonnes par jour pour produire un concentré à 30% de Cr_2O_3. La production de concentré représentera 14% de l'alimentation initiale de l'usine. Nous recommandons que le design du concentrateur soit basé sur une séparation en milieu lourd suivie par un système de concentration par gravité pour obtenir un concentré convenable avec un maximum d'efficacité et d'économie.

3.2.6 Le mode d'exploitation du dépôt de chrome doit être à ciel ouvert, en opérant séquentiellement le long de la direction du filon. L'épaisseur recommandée pour la coupe primaire est de 50 mètres. L'épaisseur pour la coupe secondaire doit être de 50 mètres supplémentaires. La pente de la fosse doit être de 55 .

3.2.7 La phase IV du programme doit être entreprise afin de préciser et développer une économie maximale pour le procédé et les infrastructures; diverses analyses de sensibilité pour voir l'effet sur le coût de production; et le marché.

3.2.8 Il est bien entendu et recommandé que les travaux suivants soient effectués sur le site préalablement aux activités de production:

a) des tranchées supplémentaires pour l'échantillonnage du dépôt
b) prélèvement d'échantillons séparés, plutôt que composites, dans la zone principale pour déterminer la localisation des minéraux et la teneur de chaque partie de la zone.
c) un programme de forage et de prélèvement de carottes selon un quadrillage
d) une usine pilote sur le site pour optimiser les procédés et la récupération du concentré.

4.0 COMMENTARY

The assumption has been made that the concentrates produced must be converted to a saleable product which is competitive in the North American market place. That is, no consideration has been given to the possible development of these substantial reserves as a strategy source of chromium due to a possible interruption of supply to North America from traditional sources; nor has any consideration been given to the economic impact of mineable values of the platinum group minerals (PGM's). Should PGM's, which are traditionally associated with chromium, be found in economically recoverable amounts, the scale of operation could be reduced and/or the overall economics substantially increased. There was no evidence that PGM's should be given further consideration from the information contained in the Phase II report.

Due to the geotechnical aspects of the deposit, it is quite apparent that the mining and concentrating costs cannot stand alone in producing a competitively priced product. Preliminary calculations suggest an underground mining cost in excess of $40 per ton of concentrate produced, which is totally unexceptable. The narrowness of the "ore" zone plus the 80° to 90° attitude of the deposit results in a high waste to ore ratio in the open-pit operation, which in turn restricts the depth of operations if the costs are to be contained within reasonable limits. This restricts the operating life of the "Chrome" deposit to a maximum of ten years. However, similar circumstances will probably apply to other deposits along the strike length.

At present, eight properties have been identified along the total strike length of approximately 58 kilometres. They represent approximately 30 kilometres of potential mineable reserves. To conform with the economic model which this preliminary reports has developed, it suggests a consortium or joint venture may be required to mine these strategic reserves at a sustained rate. The joint venture partners would then have to extend their ownership to the smelter, or if a custom smelter, they would be required to participate in the smelter returns.

A smelter for the production of stainless steel is a definite requirement. Ferrochromium ("charge chrome") is sold on the basis of contained chromium, and therefore lower grade alloys are at a competitive disadvantage relative to smelting costs and freight charges. The high carbon ferrochromium market presently trades in three grades which contain 50 to 55 percent, 56 to 64 percent, and greater than 64 percent chromium. An alloy containing only 40 percent (the indicated quality of Bird River material) would not be marketable unless it was offered at a substantial price reduction below the 50 to 55 percent grade.

During the smelting process described under "Process Selection" in Section 8 of this report, the 48 MVA submerged-arc furnace produces a ferroalloy containing 40.1% chromium. On an annual basis, the 90,620 net tons of molten alloy could be processed into 81,500 net tons of saleable alloy in various size ranges. Assuming that there is a market for this off-grade "charge chrome", the price would in all likelihood have to be in the range of 26 to 34 US cents/pound of contained chrome FOB Pittsburgh, depending on the prevailing prices of "charge chrome" itself. At a cost of approximately $60 per net ton for chromite concentrates produced from Bird River ores, the cost of producing a molten alloy is about 50 US cents/pound of contained chrome FOB Pittsburgh, having allowed 10 percent for the required additional plant costs. With a capital charge for a $100 million ferroalloy smelter of approximately 20 US cents, the total break-even cost would be about 70-75 US cents/pound of contained chrome, or over twice the market-perceived value for this material.

The major raw material for electrolytic chromium is high carbon ferrochrome containing 65 percent chromium. If a lower grade alloy is to be considered for electrolytic chromium production, the cost of disposing of a larger quantity of iron, due to the lower Cr:Fe ratio of the concentrate, would have the effect of driving down the value of this alloy even further.

The cost of producing ferrochrome from Bird River concentrates precludes considering the "charge chrome" market. Therefore, the greatest opportunity for success in commercializing the Bird River chromite ores is to manufacture stainless steel directly. The greatest potential market for stainless steel manufactured in Manitoba will be as semi-finished steel in the form of slabs. Further processing into "hot-bands" and even further to "cold-rolled" steel might also be considered.

4.0 COMMENTAIRES

Il a été assumé que le concentré obtenu doit être converti en un produit commercial compétitif pour le marché nord-américain. De fait, aucune considération n'a été accordée pour proposer le développement de ces réserves substantielles comme étant une source stratégique de chrome, advenant une éventuelle interruption de l'approvisionnement américain à partir de sources traditionnelles. De même, l'impact économique relié à l'exploitation des minerais du groupe platine n'a pas été considéré. Si des minerais du groupe platine, traditionnellement associés au chrome, étaient découverts en quantité économiquement exploitable, l'échelle des opérations pourrait être réduite et/ou les coûts d'ensemble pourraient augmenter considérablement. Les informations contenues dans le rapport sur la phase II ne démontrent pas que les minerais du groupe platine doivent être l'objet d'une étude particulière.

A cause des aspects techniques de la géologie du dépôt, il apparaît évident que les coûts d'exploitation minière et de concentration, toute autre considération exclue, ne permettent pas d'obtenir un produit à un prix compétitif. Des calculs préliminaires évaluent un coût d'exploitation minière sous-terraine excédentaire de $40 par tonne de concentré produit, ce qui est totalement inacceptable. L'étroitesse de la zone contenant le minerai de même que l'altitude du dépôt (80 à 90) concourent pour produire un rapport débris/minerai élevé pour une opération à ciel ouvert, ce qui en retour pose des restrictions quant à la profondeur d'opération si les coûts doivent être limités raisonnablement. Ceci restraint la durée d'opération du dépôt de chrome à un maximum de 9 années. Toutefois, des circonstances similaires vont probablement prévaloir pour d'autres dépôts le long de la direction du filon.

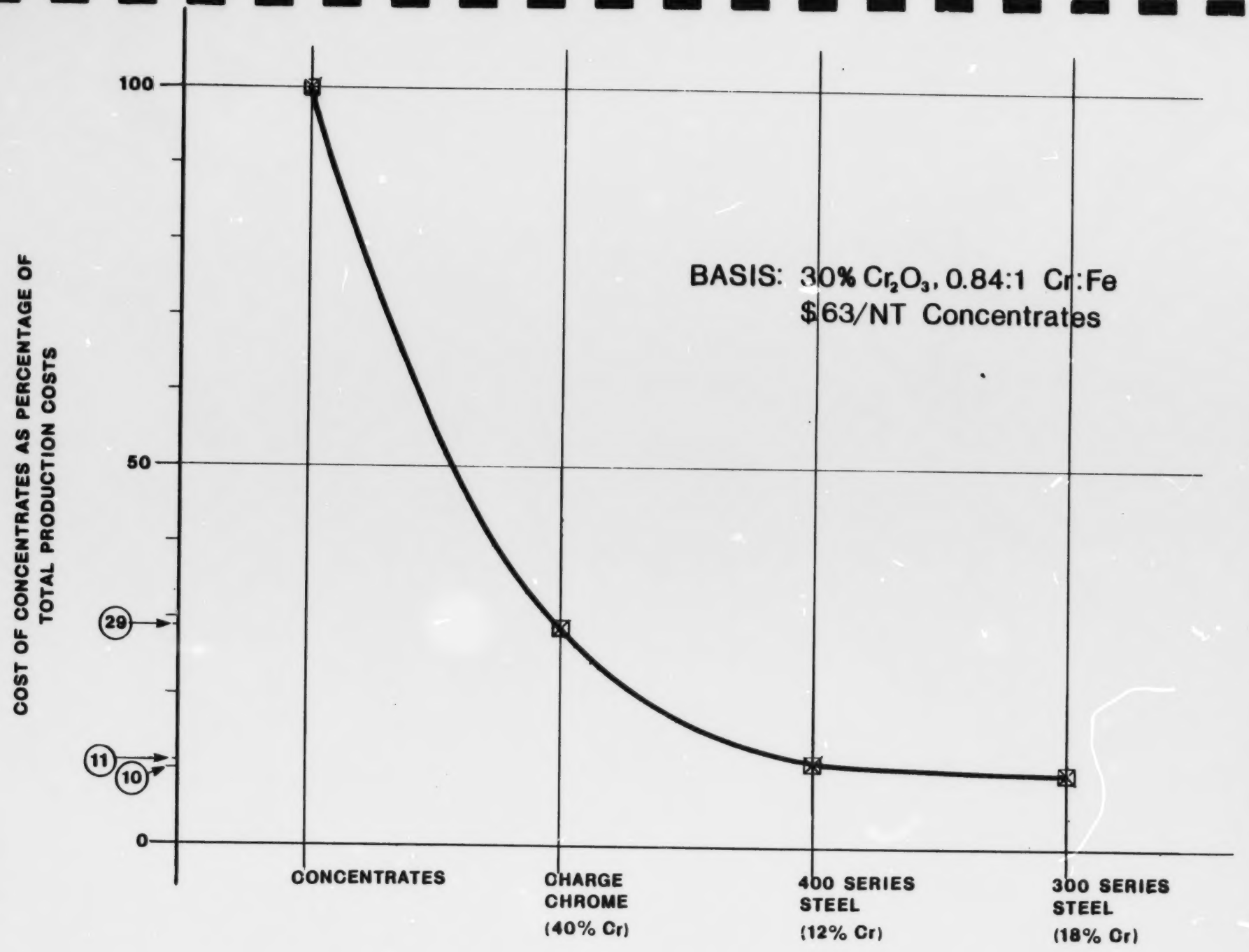
Actuellement, huit propriétés ont été identifiées le long du filon qui s'étend sur approximativement 58 kilomètres. Ceci représente environ 30 kilomètres de réserves potentiellement exploitables. Afin de respecter le modèle économique développé dans ce rapport préliminaire, il pourrait être nécessaire d'établir un consortium pour exploiter ces réserves à un régime soutenu. Les partenaires du consortium devront être propriétaires de la fonderie, ou s'il s'agit d'une fonderie produisant pour des clients, ils devront participer aux profits de la fonderie.

Une fonderie pour la production d'acier inoxydable est définitivement nécessaire. Les ferro-chromes (chrome de charge) sont vendus sur la base de leur contenu en chrome et par conséquent, les alliages à faible teneur sont désavantagés, compte-tenu des coûts pour la fonte et le transport qui sont les mêmes pour tous les alliages. Le marché des ferro-chromes à haute teneur en carbone est actuellement établi pour trois grades selon la teneur en chrome, soit de 50 à 55%, de 56 à 64% et plus de 64% de chrome. Un alliage contenant seulement 40% de chrome (qualité indiquée du matériel de la rivière Bird) ne pourrait pas être commercialisé, à moins d'être offert à un prix suffisamment réduit, inférieur au prix d'un alliage à 50-55% de chrome.

Dans le procédé de fusion, décrit dans la partie intitulée "Process Selection" de la section 8 de ce rapport, le four à arc submergé produit un ferroalliage contenant 40.1% de chrome. Sur une base annuelle, les 90 620 tonnes d'alliage fondu peuvent être traitées pour produire 81 500 tonnes d'un alliage vendable en différentes dimensions. En assumant qu'il y ait un marché pour ce chrome de charge hors-grade, le prix devrait être selon toute vraisemblance de l'ordre de $0.26 à $0.34 US par livre de contenu en chrome, FAB Pittsburgh, dépendant du prix attribué au chrome de charge. A un coût d'environ $60 par tonne de concentré de chromite, le coût de production d'un alliage fondu est environ de $0.50 US par livre de contenu en chrome, FAB Pittsburgh, les coûts additionnels de 10 % requis pour l'usine ayant été ajoutés. Une fonderie pour ferroalliage de $100 millions représentant une charge en capital d'environ $0.20 US, le point mort pour le coût total serait environ de $0.70 US par livre de contenu en chrome, soit plus de deux fois la valeur du marché pour ce produit.

La majorité des matières premières pour la production de chrome électrolytique est constituée d'alliages ferro-chrome à haute teneur en carbone, contenant 65% de chrome. Si on considère un alliage de qualité inférieure pour la production de chrome électrolytique, le fait de disposer d'une plus grande quantité de fer à cause du faible rapport Cr:Fe du concentré, aurait comme effet de diminer encore plus la valeur de cet alliage.

Le coût de production de ferro-chrome à partir des concentrés de la rivière Bird écarte la possibilité de considérer le marché du chrome de charge. Par conséquent, la production d'acier inoxydable sur place, constitue la meilleure option pour la commercialisation des minerais de chromite de la rivière Bird. Le plus grand marché potentiel pour l'acier inoxydable produit au Manitoba réside dans les produits semi-finis sous forme de brames. Le procédé de laminage à chaud ("hot bands") et même le laminage à froid de l'acier pourraient éventuellement être considérés.


COST OF CONCENTRATES AS PERCENTAGE OF TOTAL PRODUCTION COSTS
100
50
29
11
10
0
BASIS: 30% Cr_2O_3, 0.84:1 Cr:Fe
$63/NT Concentrates
CONCENTRATES
CHARGE CHROME (40% Cr)
400 SERIES STEEL (12% Cr)
300 SERIES STEEL (18% Cr)

5.0 THE "CHROME" CLAIMS - ECONOMIC GEOLOGICAL ASSESSMENT

5.1 Introduction

This section of the report summarizes the main geological considerations which have a bearing on technical and economic viability. Much of the information was derived from the reports listed in Appendix 1, and from a meeting held with Federal and Manitoba Government representatives in Ottawa on February 9, 1987.

The work calls for an assessment of economic viability based on the results of the 70 tonne sample that was tested by Lakefield Research of Canada Ltd. That sample is a composite of the two most favourable chromitite zones in the Bird River Sill together with the very low grade material in between, which represents about 75% of the total sample by weight. The Hatch study considers bulk open pit methods which would feed the mill with material represented by the 70 tonne sample.

The principal geological considerations which have an impact on economic viability at the Chrome Property are briefly described.

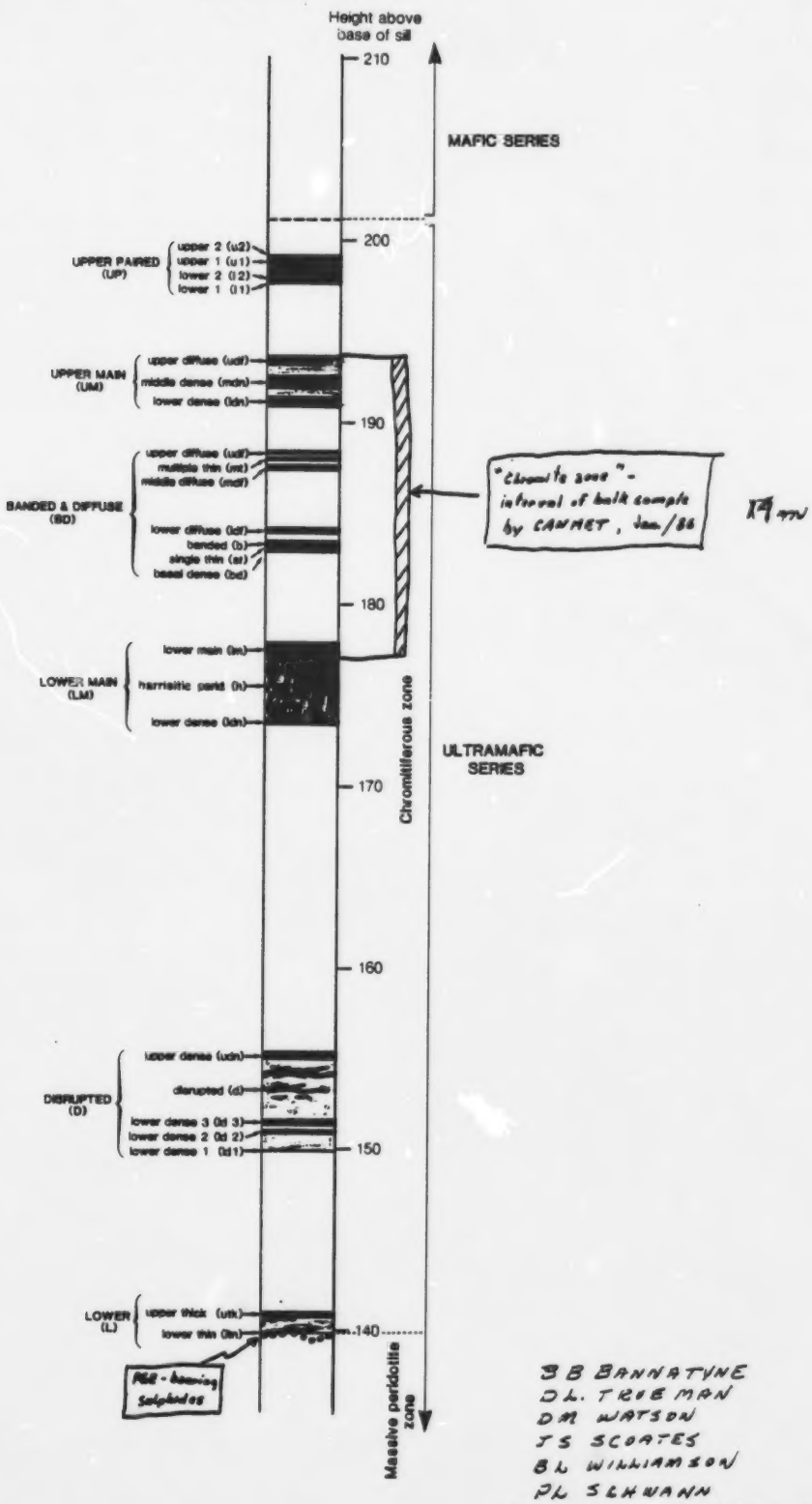
5.2 The Chrome Property

The test sample represents a 14 m section from the Upper Main Zone to the top seam in the Lower Main Zone (inclusive). For convenience, that section has been designated the "Main Zone" in this report. A perspective of the distribution of the tonnes and grade in the "Main Zone" is as follows:

Upper Main: Average width 2.59 m S.G. 3.368
Middling: Average width 10.7 m S.G. 3.0 (assumed)
Lower Main: Average width 0.64 m S.G. 3.4

Upper Main: Tonnes/vert m/strike m = 8.723 @ 18.2% Cr_2O_3
2500 m strike = 21,807 tonnes/vt m
Middling: Tonnes/vert m/strike m = 3.21 @ 2.74% Cr_2O_3
2500 m strike = 80,250 tonnes/vt m
Lower Main: Tonnes/vert m/strike m = 2.176 @ 23.0% Cr_2O_3
2500 m strike = 5,440 tonnes/vt m

CHROMITIFEROUS ZONE STRATIGRAPHY



SB BANNATYNE
DL. TRUEMAN
DM WATSON
JS SCOATES
BL WILLIAMSON
PL SLAWANN
- MANITOBA ENERGY + MINES GEOLOGICAL SERVICES
- GEOLOGICAL SURVEY of CANADA

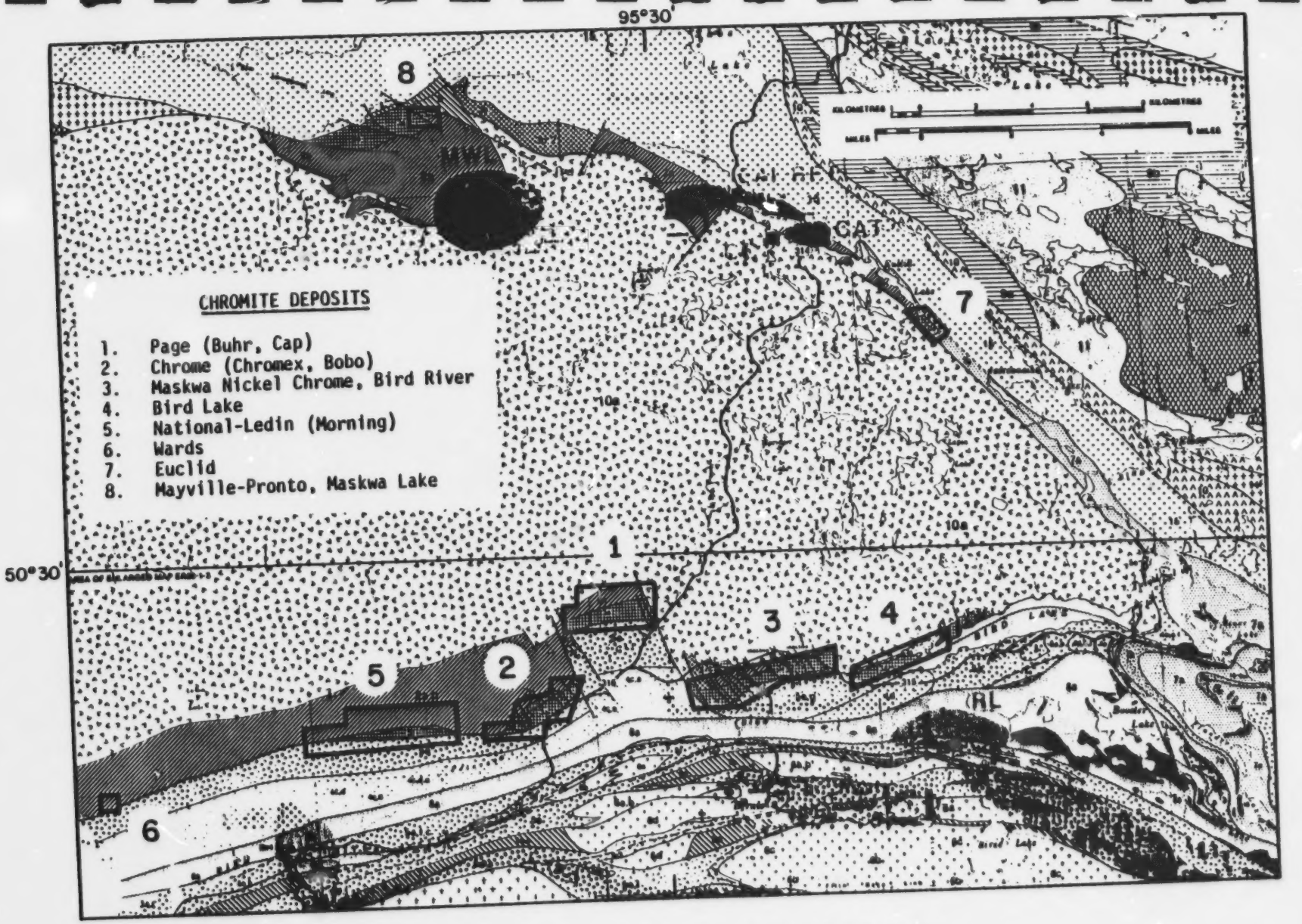


Figure 2. Geology of the Bird River-Cat Lake area; and location of chromite deposits (from Černý et al., 1981, Map ER80-1-1)

Combined

Main Zone: Tonnes/vert m/strike m = 43 @ 6.9% Cr_2O_3

2500 m strike = 107,497 tonnes/vt m

There is no available evidence to suggest that chromitite and PGM zones beyond the Main Zone would make a significant contribution to economic viability.

There are numerous transverse faults. Wallrock dilution is taken to be equal to the "loss of ground" by faulting. A millhead grade of 6.9% Cr_2O_3 is retained to conform with the Lakefield sample.

5.3 Geological Setting

The Bird River Sill is a layered ultramafic to mafic intrusive complex, averaging about 1000 m thick. It forms an eastward plunging anticline with both limbs dipping steeply at 60 to 90 degrees. The south and north limbs are 32 km and 24 km long respectively. Continuity is interrupted by faulting and younger granite intrusives, so that the total length of the outcrop and sub-outcrop beneath overburden and water may not exceed 30 km.

The 70 tonne sample sent to Lakefield for beneficiation tests was taken from the Chrome Claims on the southern limb. In this report, the geological characteristics which influence mine design and beneficiation relate to those occurring at the Chrome Property. However, there are apparently very close similarities to the nearby properties, and many similarities to the properties on the north limb. The significance of this is that shallow mining and/or selective procedures may require mineral reserves beyond the Chrome Property for optimum economic viability. Furthermore, combining reserves would have a favourable impact on capital and operating costs relating to infrastructure, plant, services and administration.

5.4 The Chromitiferous Zone

The Chromitiferous Zone is about 60 m thick and occurs near the top of the ultramafic portion of the Bird River Sill. Chromite occurs as numerous, narrow chromitite layers, disseminated layers and scattered grains within intercalations of serpentized picrite, peridotite and dunite. The most superior concentrations of chromite occur within two zones that are designated Upper Main Zone and Lower Main Zone. The Upper Main Zone (average width 2.59 m) consists of three dense, closely-spaced chromitite seams. The Lower Main Zone has most of its chromite concentrated in a dense seam (average width 0.64 m) at the top of the zone.

The Lakefield bulk sample represents a cross-section from the uppermost seam in the Upper Main Zone to the upper seam in the Lower Main Zone (inclusive). That section appears to have the best potential for a relatively wide in-situ ore zone. For convenience, it will be referred to as the "Main Zone" in this report.

Although a pit outline may encompass narrow chromite seams and disseminations outside the Main Zone (including a thin band containing platinum group minerals bearing sulphides - PGM's - about 30 m in the footwall), there is no available evidence to suggest that any of that material would make a significant contribution to economic viability. The PGM potential needs to be investigated further.

The Main Zone is approximately 14 m wide. For economic purposes, the Main Zone can be conveniently divided into the relatively rich Upper Main Zone (averaging width 2.59 m), the rich Lower Main Zone (average width 0.64 m), and in between, the low grade "Middling Zone" (average width 10.7 m). The Middling Zone consists of the weakly-mineralized Banded and Diffuse Zone which is flanked on both sides by virtually barren rocks.

5.5 Mining Geology

Since the Lakefield sample is a composite of all the material in the Main Zone, it is assumed that the entire 14 m zone would be bulk-mined and milled. The geological implications are:

(a) The Lakefield sample grade is 6.9% Cr_2O_3. If the sample has been excavated cleanly from the footwall to the hangingwall, then that can be considered an in-situ grade. In practice, the in-situ grade would be diluted by an amount related to the effects of layering, faulting, wallrock quality, pit geometry and grade control disciplines. There is a significant amount of transverse faulting with major blocks having strikes of 30 m to 500 m. That will influence pit design and have an influence on dilution.

(b) An order-of-magnitude dilution factor of 10% might be appropriate, but since a "loss of ground" by faulting is anticipated, it is suggested that diluted tonnage = loss of ground for this exercise. Furthermore, since some selective breaking is always possible, it is suggested that a millhead grade of 7% Cr_2O_3 be retained for the study, so as to conform with the beneficiation tests done to date.

(c) The potential in-situ mineral reserve at the Chrome Property for the Main Zone averaging 14 m wide, is 43 tonnes/vert. m/strike m at the grade of 7% Cr_2O_3.

(d) Reports indicate that Cr_2O_3 grades from diamond drilling are lower than those from trenching. That might be explained by either poor recoveries in boreholes due to friable chromite and a soft matrix, or by biased trench sampling (too much chromite reporting in the samples) or both. Similar problems exist in the Bushveld where large diameter boreholes and special core barrels are used to improve sample recovery.

6.0 MINING OF THE "CHROME" DEPOSIT

6.1 General Considerations

The mining information and cost estimates provided in this report are largely based on current Canadian operation experience. The cost estimates were developed from first principles and verified against the experience of two small open pit operations in Northern Ontario, operations producing approximately one million tons of ore per year with low stripping ratios. The cost accuracy in the report is significantly influenced by the lack of an on the site visit, virtually no geological drill hole and assay data, and regional rather than local geological tectonic data for the sampled area of the Chrome Deposit.

Access to the mine site, site preparation and capital cost estimates for the selected mining rate are meant to be used as first order estimates.

The potential ore-body of 14 m in width is narrow for conventional open pit mining. For open pit mining it is approximated as a slice with a total length of 2,500 meters, 14 meters wide and a depth of 100 meters. Mineable tonnage and pit life based on a mill capacity of 1,225,000 MT per annum are as follows:

Pit Depth	Ore (MT)	Waste*	Ore Life (yrs)	S/R**
50 m	5,372,500	24,890,000	4.38	4.61
100 m	10,745,000	76,806,000	8.77	8.15

* 14 m of internal waste included

** Stripping Ratio $= \frac{\text{Waste + Ore}}{\text{Ore}}$

6.2 Mining Programs and Costs

Geological considerations for mining indicate that selective mining for the relatively narrow and high-grade chromitite seams contained in the sampled main zone will be extremely expensive and difficult, due to weak ground. Their economic viability can be greatly enhanced with serious consideration for waste sorting after mining and before milling. From the available

geological information on the Chrome Deposit, the expected ore potential is estimated in excess of 10 million tonnes, containing 7% Cr_2O_3, to a 100 meter depth. Extension of ore potential is open to depth.

The present ore potential is amenable to open pit mining to the 50 meter and 100 meter horizon below surface.

The viable pit life approaches, at best 9 years and there is some degree of optimism to believe that underground mining would extend mine life beyond 10 years.

The mining operation as planned for a 50 meter pit and/or a 100 meter pit, would produce 1,225,000 tonnes of ore per year. This will require a total of 67 people, including 15 professional, 38 production personnel and 14 qualified tradesmen.

For strategic planning purposes the estimated mine operating costs for the selected and alternative mining programs are estimated as follows:

Pit Depth	$/Tonne	Life (Yrs)	Comments
50 m	6.22	5	2,500 m of strike length required every 4 years for sustained operations.
100 m	9.40	9	
beyond 100 m	15.00	+10	transition to underground operations.

Open Pit Mine Capital Costs

Requirement are estimated as follows:

Over burden stripping	$ 965,000
Major Pit Equipment	16,000,000
Auxiliary Pit Equipment	3,000,000
Mine/Office/Warehouse/Dry and Repair Bays	4,500,000
Primary Crushing/Dust/Screen	3,000,000
Miscellaneous Pumphouse/Electrical	2,000,000
SUB-TOTAL	$ 29,465,000
Contingency @ 20%	5,893,000
TOTAL	$ 35,450,000

6.3 Geological Considerations for Mining

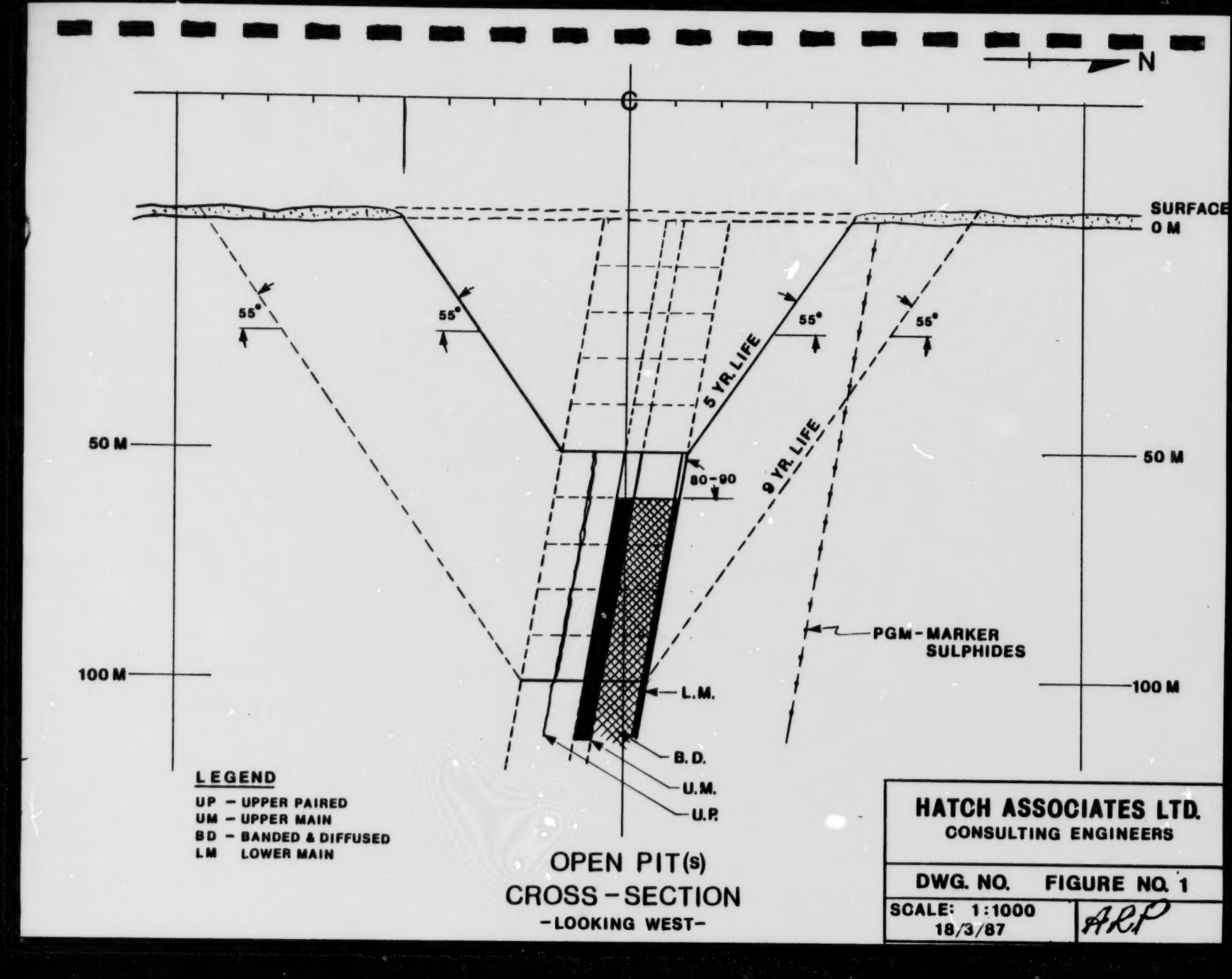
The main zone, as a composite sample, is 14 meters wide containing an average grade of 7% Cr_2O_3. As noted in Section 5 the Main Zone can be further divided into two distinct veins of higher grade material as follows:

- The Upper Main Zone with an average width of 2.59 meters consists of three closely spaced chromitite seams containing 18.2% Cr_2O_3.

- The Lower Main Zone with an average width of 0.64 meters consists of one dense chromitite seam containing 23% Cr_2O_3.

- The Middle Banded and Diffused Zone with an average width of 10.7 meters containing 2.74% Cr_2O_3.

The immediate serpentine and olivine materials are weak host rocks associated with the high grade chromitite veins and there are a number of transverse faults. The significance of these two narrow high-grade chromitite seams separated by the low grade middle zone is that individually they would be extremely difficult and costly to mine selectively with small capacity equipment and relatively high labour costs. Their economic viability might be enhanced greatly if future consideration is seriously given to internal waste sorting after open pit or bulk underground mining and before milling.


N
SURFACE
0 M
55°
55°
55°
55°
5 YR. LIFE
9 YR. LIFE
50 M
50 M
80-90
PGM-MARKER
SULPHIDES
100 M
100 M
L.M.
B.D.
U.M.
U.P.
LEGEND
UP – UPPER PAIRED
UM – UPPER MAIN
BD – BANDED & DIFFUSED
LM LOWER MAIN
OPEN PIT(S)
CROSS-SECTION
-LOOKING WEST-
HATCH ASSOCIATES LTD.
CONSULTING ENGINEERS
DWG. NO. FIGURE NO. 1
SCALE: 1:1000
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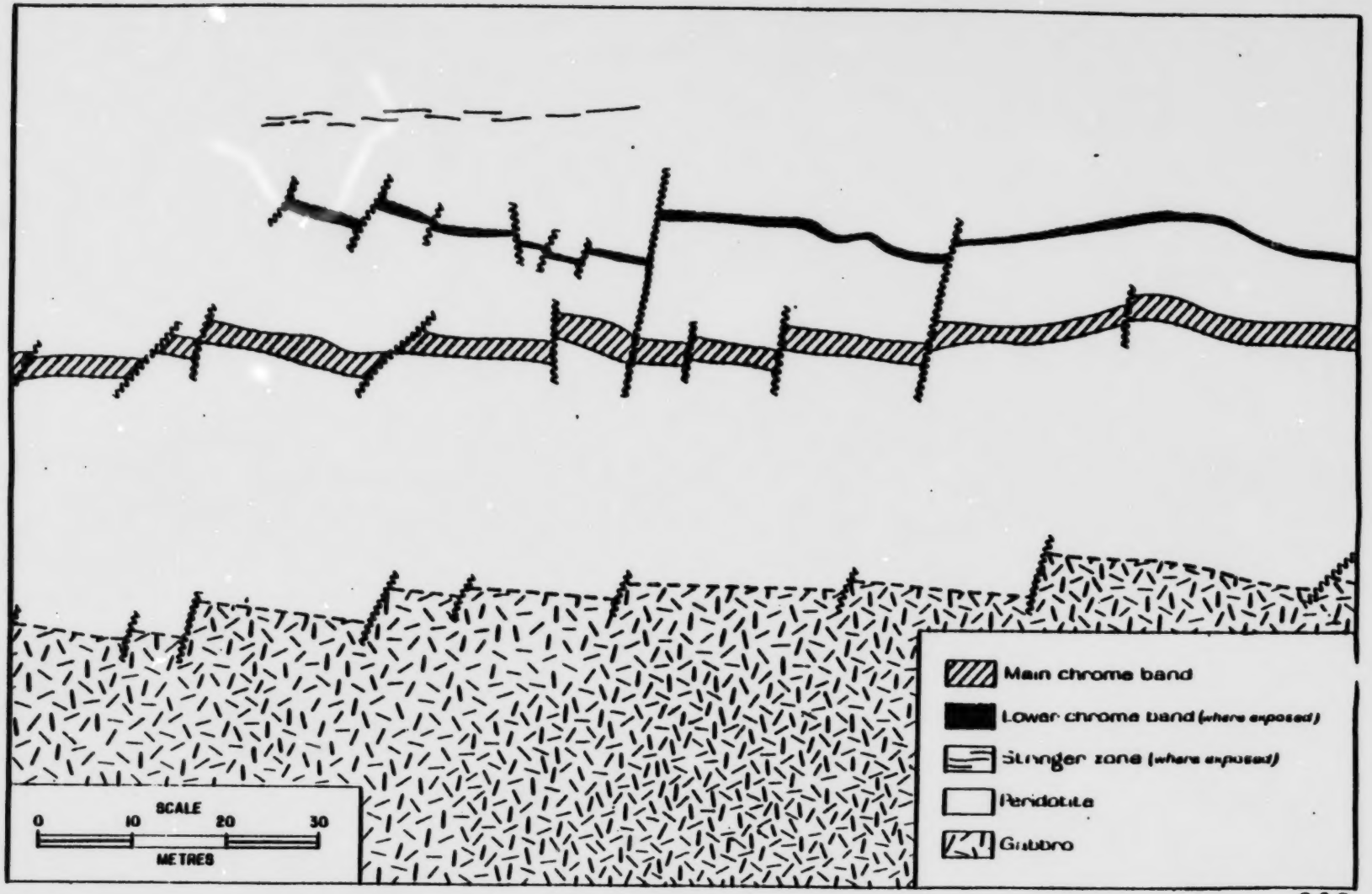


Plan of part of chromite zone on Chrome property (from Bateman, 1943). AFTER B.B. BANNATYNE/D.L. TRUEMAN. 1982 - REPRINT 1986

FIGURE 2

6.4 Mine Design

Figure 1, shows a typical cross section of the Chrome Deposit with a planned open pit slope of 55° and a depth of 50 and 100 meters. The deposit has a number of transverse displacements of ore on strike due to cross-faulting. (Figure 2). Three or more pit areas would be working simultaneously. Hence pit(s) deepening is limited to the selected bench height of 10 meters, in order to insure grade control. Pit deepening and maximum advance per year would cover the entire strike length. Pit(s) deepening is determined by the required mill capacity of 1,225,000 tonnes divided by the expected tonnage per ore bench - 1,225,000 divided by 1,074,500 or 1.14 benches per year.

From each loading place it must be possible to load and transport ore and the waste rock. This means the waste rock and ore is removed with a working slope angle of 30° in each pit. With the narrow ore width of 14 meters the working face requires a mining front in excess of 18.5 meters for a (31.8 MT) 35 ton truck, 23.5 meters (45.4 MT) 50 ton truck and 26.8 meters for a (77.1 MT) 85 ton truck. The break even pit cost versus underground mining cost is estimated to be $15 per tonne. The operating cost for mining the Chrome Deposit beyond the 100 meters depth, say 10 years in the future, is uncertain at this time. A great deal of detailed pit planning is required to determine exactly when ore liberation and waste rock removal should take place in any given year. These detailed open pit activities are beyond the scope of this study report. Therefore, it is evident for this study that the mining front for ore must approach 30 meters. Hence the typical cross section of the pit shows a planned mining width of 28 meters, and a pit slope of 55°. With a 28 meter mining front open pit mining is practical to the 50 meter depth and it is technically feasible to the 100 meter depth.

6.5 Section of Mine Production Rate/Method/Equipment

The ore potential is estimated at 10,750,000 MT from a 2,500 m strike length, with a 14 m ore width (inclined at 80 - 90°), to a depth of 100 m. The specific gravity of the ore is estimated at 3.07 - 3.0. The expected pit slope is 55°. The mined grade is estimated at 7% Cr_2O_3. Assumed ore losses of 10 - 12% due to geological faulting, are off-set by the estimated waste dilution of 10 - 12%. The selected mine production rate for ore

of 3,500 TPOD (tonnes per operating day) or 1,225,000 T/Y, is deemed optimistic for a 14 meter ore front. It is at best obtainable by widening the mining front to 28 meters in order to accommodate a pit working slope of 30° for conventional pit equipment. The removal of the 14 meters of internal waste with the widened mining front of 28 m has been included into the waste removal tonnages as shown in Table 6.3. As confirmed from operations in Northern Ontario, the presently established payroll costs for labour and tradesmen are generous and onerous. The estimated manpower costs are based on existing Northern Ontario operations, which are shown in Table 6.1. Tables 6.4 and 6.5 show production tonnages of ore and waste and estimated operating costs for the alternative pit depths. Table 6.2 shows the production capacity of the selected equipment. Table 6.6 summarizes the estimated unit operating costs per tonne of material.

The required ore production of 3,500 tonnes per day on a 350 day/year operating schedule is deemed feasible for shallow pits of 50 m in depth. Several replacement pits outside the sampled chrome deposit are deemed necessary to sustain production beyond 4 years at a 1,225,000 T/Y mill capacity. For the present study the 14 m ore thickness and an average grade of 7% Cr_2O_3, are assumed to be uniform to the 100 m depth below surface. Extraction of ore beyond 50 m is also feasible with a higher stripping ratio for a 100 m pit depth and with the concept of in-the-wall haulage systems for pit mining.

The concept of in-the-wall pit haulage when compared to conventional in-the-pit haulage ramps is as follows: where ground permits, single or twin ramps are driven in the pit walls. The ramp(s) are used with cross-cuts at 20 m interval connections to the working pit levels. Economic justification is compared to the waste excavation cost for both methods.

The selection of pit rotary or in-the-hole drilling, loading and hauling equipment are fully compatible today with small to medium underground equipment. Hence, future transition to an underground operation below the 100 m horizon is also within practical feasibility for the Chrome deposit. These alternative studies require more geological data to ensure stable ground conditions. For these reasons they are presently deemed beyond the scope of this preliminary study.

TABLE 6.1

MANPOWER*

MINE ADMINISTRATION				PAYROLL $/YEAR
Production Manager	1			110,000
Chief Engineer	1			100,000
Chief Geologist	1			90,000
Engineer/Geologist	2		85,000	170,000
Engineering Tech.	2		60,000	120,000
Geologist Tech.	2		60,000	120,000
Warehouse	2		50,000	100,000
Clerks	4		40,000	160,000
TOTAL	15			$970,000
PRODUCTION MANPOWER		$/Hr. Base Wages		
Mine Foreman	2		90,000	180,000
Drillers	6	16	86,240	517,440
Loaders	6	14	75,460	452,760
Truckers	10	13	70,070	700,700
Dozer/Grader	4	14	75,460	301,840
Blasters	4	14	75,460	301,840
Labours	6	12	64,680	388,080
TOTAL	38			$2,742,660
MAINTENANCE/SERVICE				
Foreman	2		90,000	180,000
Mechanics	2	16.50	88,935	177,870
Electricians	2	16.50	88,935	177,870
Hydraulics	2	16.50	88,935	177,870
Mechanic Helpers	2	14.50	78,155	156,360
Clerks	2	12.00	64,680	129,360
Miscellaneous	2	12.00	64,680	129,360
TOTAL	14			$1,128,690
GRAND TOTAL	67			$4,841,350

* Includes 40% fringe benefits and 10% over-time/shift premiums

TABLE 6.2

REQUIREMENTS FOR 50 & 100 m PITS			PIT EQUIPMENT - PRODUCTION CAPACITY					
EQUIPMENT	QUANTITY		Tonnes Per Oper. Shift	% Mech. Availability	% Use of AVail. Time	Utilization (%)	Average Tonnes/ Shift	Tonnes/ Year* (000)
	100 m Pit	50 m Pit						
Rotary Drill 9 7/8"	1	1	11,820	75	70	53	6,206	5,585
D.T.H./ Top Ham.	2	2	8,000	75	70	52.5	4,000	3,600
Elect-Hydrlc. Shovel	1	1	8,360	80	85	68	5,685	5,117
Front-End Loader	2	2	6,000	80	85	68	4,080	2,856
Truck 85-77.1*	8	6	2,450	80	90	72	1,764	1,235
50-45.4	13	10	1,445	80	90	72	1,040	728
35-32	19	14	1,018	80	90	72	733	513

* 700 Operating Shifts Per Year.

TABLE 6.3

OVERBURDEN STRIPPING

The ore body is approximated as a slice with a length of 2,500 m, a width of 14 m and pit depths of 50 and 100 m. For all pit depths the slope of the overburden has been approximated at 45° - .79 radius.

The overburden tonnage, with a SG of 2 and a thickenss of 3 meters, over the entire length of the ore potential is estimated to contain 715,000 tonnes as follows:

Over the ore		210,000
Along the ore		420,000
At the ends		8,400
At the corners		47,100
Over the ramp		30,000
	TOTAL	715,000

WASTE REMOVAL

Assume ore is 14 m and associated waste 14 m for 28 m mining front.

Total - Ore & Waste = 28 m

PIT DEPTH: 50 m
Amount of Waste
Pit slope angle 55° d = 50 m/100 m

Along the Ore		12,250,000
At the Ends		6,860,000
At the Corners		179,594
For the Ramp		700,000
	SUB-TOTAL	19,989,594
Internal Waste		4,900,000
	TOTAL	24,889,594
Ore		5,372,500
	GRAND TOTAL	30,262,094

PIT DEPTH: 100 m

Along the Ore		49,000,000
At the Ends		13,720,000
At the Corners		1,436,026
For the Ramp		2,800,000
	SUB-TOTAL	67,006,026
Internal Waste		9,800,000
	GRAND TOTAL	76,806,026

TABLE 6.4

PRODUCTION TONNAGES OPERATING COSTS

100 ■ PIT

PROJECT YEAR	PRODUCTION TONNAGE (000) KT				STRIP RATIO	OPERATING COSTS $		
	OVER-BURDEN	WASTE	ORE	TOTAL ORE/WASTE	S/R	$/TONNE MATERIAL	TOTAL $/ ANNUM	$/TONNE OF ORE
1	715	-	45	760	-	1.15	874	N/A
2	-	8758	1225	9983	8.15	1.15	11,480	9.40
3		8758	1225	9983	8.15	1.15	11,480	9.40
4		8758	1225	9983	8.15	1.15	11,480	9.40
5		8758	1225	9983	8.15	1.15	11,480	9.40
6		8758	1225	9983	8.15	1.15	11,480	9.40
7		8758	1225	9983	8.15	1.15	11,480	9.40
8		8758	1225	9983	8.15	1.15	11,480	9.40
9		8758	1225	9983	8.15	1.15	11,480	9.40
10	715	6744	900	7644	8.49	1.15	8,900	9.40
TOTAL	715	76,806	10,745	87,551	8.15	1.15	100,684	9.40

$S/R = \frac{\text{WASTE + ORE}}{\text{ORE}}$

WASTE DILUTION - 10-12%

ORE GRADE = 7% Cr_2O_3

TABLE 6.5

PRODUCTION TONNAGES OPERATING COSTS

50 m PIT

PROJECT YEAR	PRODUCTION TONNAGE ($000) KT					OPERATING COSTS ($000)		
	OVER-BURDEN	WASTE	ORE	TOTAL ORE/WASTE	S/R	$/TONNE MATERIAL	TOTAL $/ ANNUM	$/TONNE OF ORE
1	715	2178	473	2651	5.60	1.35	3,579	7.57
2	-	5678	1225	6903	5.64	1.35	9,319	7.60
3		5678	1225	6903	5.64	1.35	9,319	7.60
4		5678	1225	6903	5.64	1.35	9,319	7.60
5		5678	1225	6903	5.64	1.35	9,319	7.60
SUB-TOTAL	715	24,890	5372	30262	5.63	1.35	9319	7.60
6	-	-	1225	-		1.35	1654	1.35
TOTAL	715	24,890	6897	31787	4.61	1.35	42,912	6.22

$$S/R = \frac{\text{Waste + ORE}}{\text{Ore}}$$

TABLE 6.6

SUMMARY OF PIT OPERATING COSTS PER MT OF MATERIAL

TONNES X 10^6 ANNUM	DRILL	BLAST	LOAD	HAUL	AUX. EQUIP	GEN. MNTCE.	GEN. ADMIN.	TOTAL
6,903	.05 .08 .13	.04 .06 .10	.07 .08 .15	.10 .13 .23	.11 .13 .24	.13 .20 .33	.14 .03 .17	1.35
9,983	.05 .08 .13	.03 .06 .09	.04 .07 .11	.07 .13 .20	.10 .09 .19	.12 .20 .32	.10 .01 .11	1.15

DISTRIBUTION WITHIN ABOVE FROM:

	50 m Pit	100 m Pit
Operating Labour	0.31	0.29
Maintenance Labour	0.17	0.11
Parts & Consumables	0.62	0.65
Mine Administration	0.14	0.10
	1.35	1.15

7.0 CONCENTRATION OF BIRD RIVER CHROMITE

7.1 Introduction

In May 1986, a railcar containing a 70 ton sample of Bird River Chromite taken from the "Chrome" deposit was received at Lakefield Research.

On December 10, 1986 Lakefield issued their final report entitled "Mineral Processing Investigations of Bird River Chromite Ore", Project No. L.R. 3122. Throughout the project, which included laboratory and pilot plant testwork, the results were discussed with the Scientific Authority, Mr. P. Andrews.

Review of the metallurgical testwork and the capital and operating cost estimates contained herein are based on this Lakefield Final Report.

7.2 Laboratory Testwork

The first phase of the Lakefield Research investigations was carried out on a laboratory or bench-scale program with a view to developing the flowsheets to be utilized in the pilot plant operation. Variations to the technical proposal were made in consultation with the Scientific Authority as the work proceeded.

7.3 Pilot Plant Operation

The second phase of the Lakefield Research investigations consisted of applying the flows developed in the laboratory to the pilot plant at a continuous feed rate of 75 kilograms per hour.

7.4 Concentrator Design

Our review of the various methods tested indicated that the preferred method of treatment would be the production of a preconcentrate by heavy media. This preconcentration in the pilot plant was effected by the use of a tri-flo TFS 150 separator. The heavy media accept product was not further treated on a pilot plant scale but various flowsheets were tested on a laboratory scale by Lakefield Research using a composite sample of the accept products.

A meeting was held at Lakefield on February 17, 1987 with K.W. Sarbutt, Manager - Minerals Processing, and I. Jackman, Project Metallurgist. These discussions confirmed our belief that for present purposes the flowsheet accepted should include heavy media separation followed by gravity separation only. Laboratory flotation and magnetic concentration of gravity separation products did not demonstrate any economic contribution to the overall efficiency of the flowsheet. This decision suggested the following flowsheet and incorporates the assumption that the final concentrate can be achieved by regrinding both middlings from the rougher and cleaner gravity separation circuits.

The pilot plant results accepted for purposes of this report are tabulated as follows (from Lakefield Report, P.312, Table 20):

Preconcentration Accept Concentrate			Chromite Concentrate			
WT.%	%CR_2O_3	%REC'Y	O'ALL WT.%	%CR_2O_3	%O'ALL REC'Y	CR/FE
50.3	11.7	89.7	14.4	30.0	67.4	0.84

7.5 Capital Cost Estimate

Based upon a feed rate of 3,760 net tons per calendar day, the battery limits estimated capital cost of the concentrator has been developed as $12,900,000.

This estimate is not the result of engineering design of the facilities; rather, the capital cost has been developed from the equipment cost of the major items of equipment required and factoring to provide the estimate of the installed plant. Many assumptions were necessary. However, the cost of major equipment items such as grinding mills, screens, heavy media and gravity separation units were solicited from equipment suppliers. Therefore, we believe the estimate to be within the limits usually considered acceptable for prefeasibility studies.

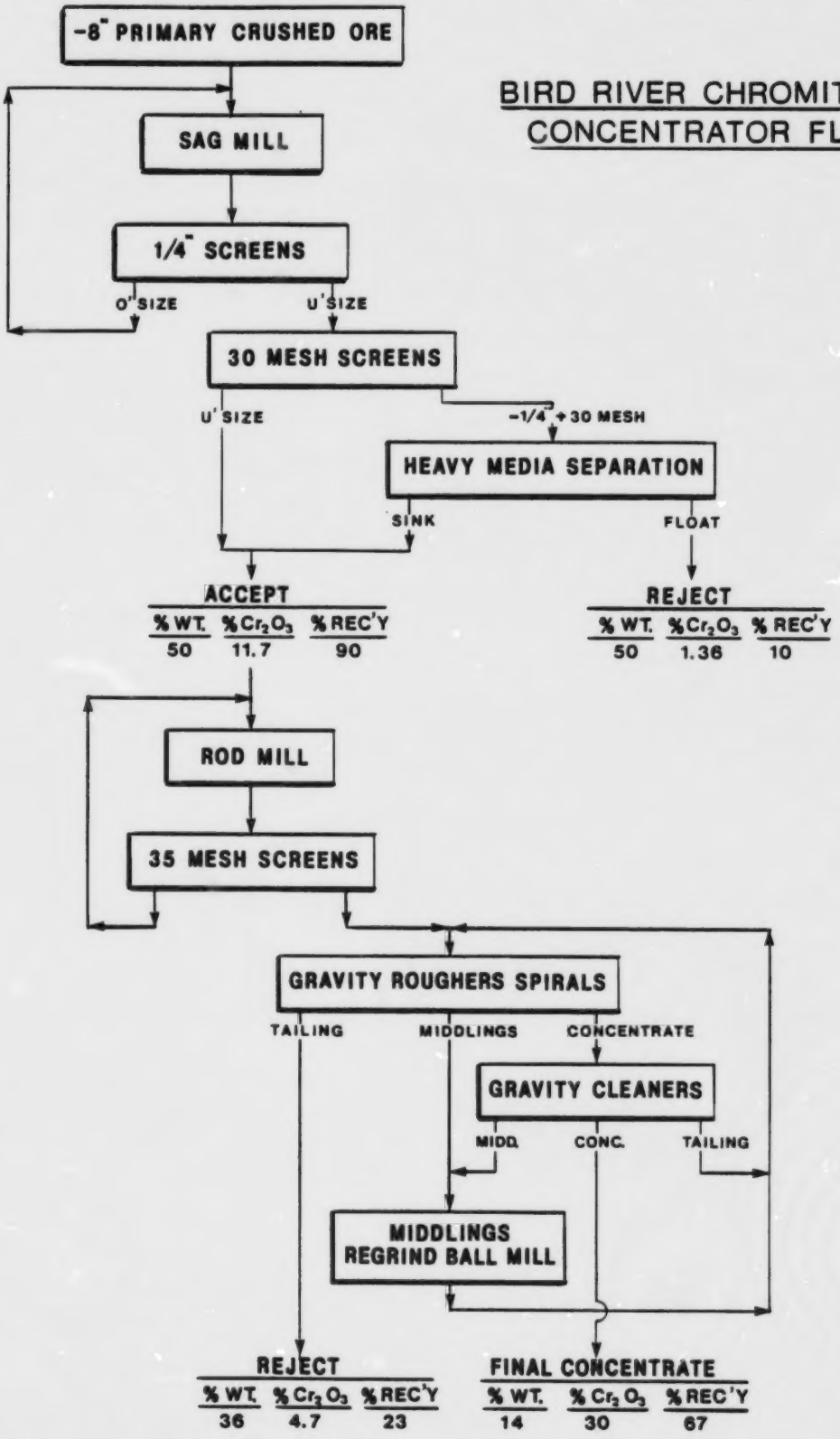
7.6 Operational Cost Estimate

The concentrator operating costs have been estimated as $3.19 per net ton of ore milled, or $22.78 per net ton of 30% chromite concentrate produced.

The operating costs have been developed from information contained in our files and conversations with those familiar with staffing and supply costs in the area.

As in the case of the capital cost estimate, the operating cost estimate is believed to be within the limits usually acceptable for prefeasibility studies.

BIRD RIVER CHROMITE PROJECT
CONCENTRATOR FLOWSHEET



8.0 THE PRODUCTION OF STAINLESS STEEL

8.1 Process Description

The feed material to the smelting operation is a chromite concentrate which reports from the adjacent mine concentrator in slurry form. As shown in Figure 1, this slurry is dewatered to 8 percent moisture, mixed with bentonite and coke fines and pelletized. The green pellets are sintered, cooled and transported to covered storage. The batching plant mixes the pellets in a given ratio with reductants such as coke and possibly coal, as well as fluxes which include quartz, dolomite and limestone. The batched material is smelted in a 48 MVA submerged-arc furnace of standard design employing Soderberg (self-baking) electrodes. The "molten ferrochromium alloy" is transferred to one of two induction furnaces rated at 15 MW each where the chromium content is diluted by the addition of mild steel scrap. The resulting "transfer metal" is charged to the Argon-Oxygen Decarburization (AOD) vessel where the carbon is removed to a low level. The "blown metal" is stainless steel which is transferred to the casting machine for production of slabs.

Furnace gas contains approximately 85 percent carbon monoxide and is therefore a medium-value calorific gas. The furnace off-gas is scrubbed to remove dust and fume. The pressure is then raised as the gas is transferred to a gas holder. This gas will be sufficient to supply all of the energy needed for sintering, ladle drying, and other sundry demands.

Table 3 lists the intermediate and final product quantities and their processing rates while Table 4 gives the metal analyses of some of these products. It will be evident that the 40.1% chromium containing ferroalloy is diluted by scrap and alloying additions to produce either 300 or 400 series steels. Table 5 lists the annual material requirements to produce 200,000 NTPA of stainless slabs. The calculations made to support the data in these tables will be found in the Appendix.

BIRD RIVER CHROMITE

FIGURE 1

PROCESS FLOWSHEET: FILTER CAKE TO STAINLESS STEEL

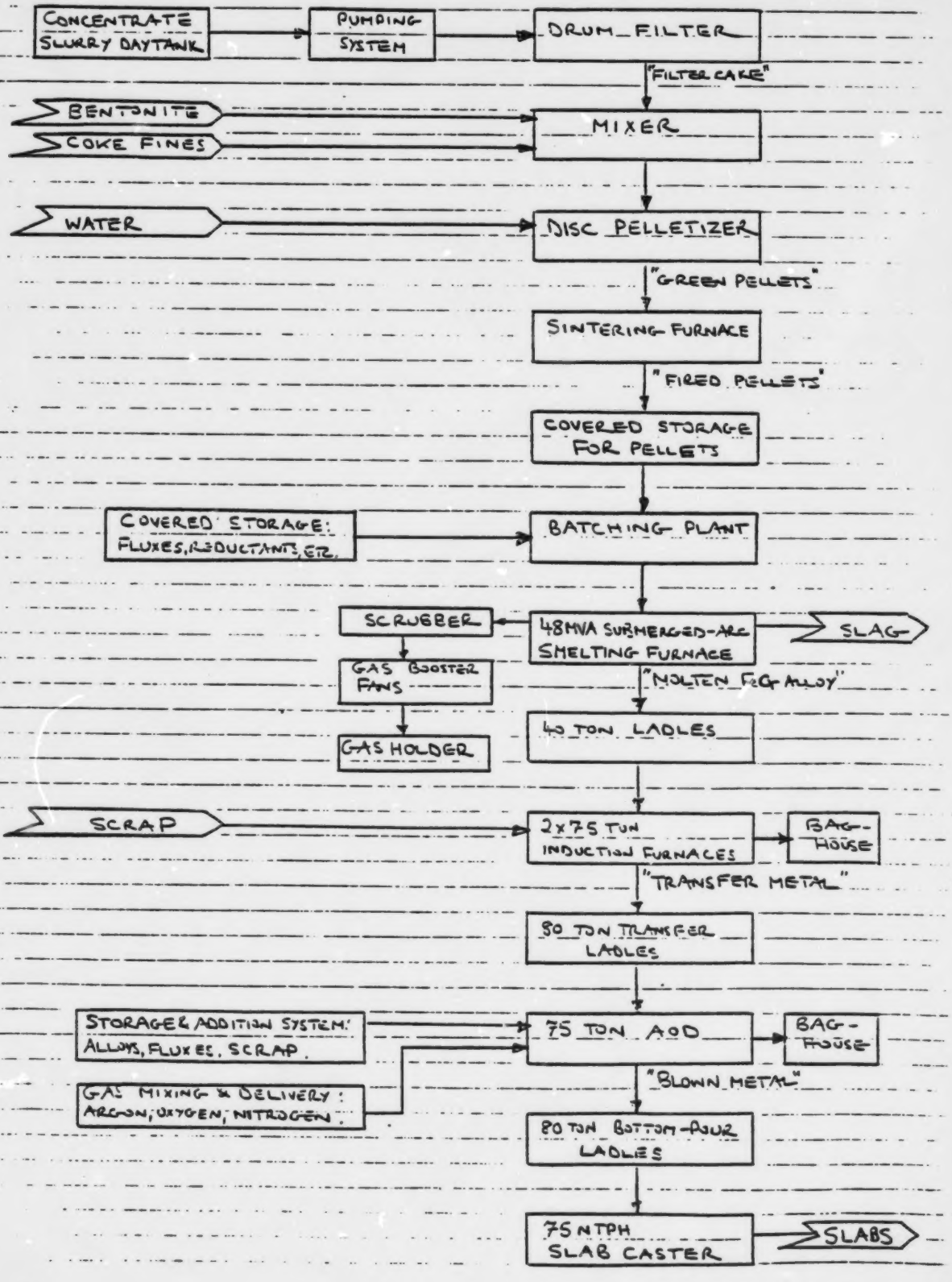


TABLE 3

PRODUCTS AND PROCESSING RATES

PRODUCTS	ANNUAL NET TONS	PRODUCTION/CONSUMPTION RATE
Concentrates	192,000	25.8 NTPH
Mixed Material	213,120	28.6 NTPH (Mixer)
Green Pellets	229,340	30.8 NTPH (Pelletizer)
Fired Pellets	914,940	26.2 NTPH (Sintering Furnace)
Molten FeCr Alloy	90,620	11.1 NTPH (Smelting Furnace)
Slag	99,690	12.2 NTPH (Smelting Furnace)
300 Series Transfer Metal	132,050	3 hours/cycle (Induction Furnace)
400 Series Transfer Metal	72,010	2 hours/cycle (Induction Furnace)
300 Series Blown Metal	138,000	83 mins/cycle (AOD)
400 Series Blown Metal	70,000	71 mins/cycle (AOD)
300 Series Slabs	150,000	54 mins/cycle (Caster)
400 Series Slabs	50,000	55 mins/cycle (Caster)

TABLE 4

METAL ANALYSES

	Cr (%)	Fe (%)	Si (%)	C (%)	Ni (%)
Molten FeCr Alloy	40.1	50.7	1.0	8.2	-
300 Series Transfer Metal	20.6	73.7	0.8	4.5	-
400 Series Transfer Metal	12.6	83.1	0.7	2.9	-
300 Series Blown Metal	18.2	BAL.	0.5	0.08	8.0
400 Series Blown Metal	11.9	BAL.	0.5	0.15	-

TABLE 5

ANNUAL MATERIAL REQUIREMENTS

Chromite Concentrates	192,000 NT
Bentonite	2,000 NT
Coke Fines	4,000 NT
Lump Coke	47,000 NT
Quartz	30,000 NT
Dolomite/Limestone	66,000 NT
Electrode Paste	900 NT
Plain Carbon Steel Scrap	114,000 NT
Electrolytic Nickel	22.5×10^6 pounds
Oxygen	283.9×10^6 SCF
Nitrogen	16.4×10^6 SCF
Argon	88.5×10^6 SCF
Lime	9,400 NT
Fluorspar	1,100 NT
FeSi 75	2,400 NT
SiMn	3,600 NT
Electric Power	402,022 MWh

8.2 Capital Cost

The process flowsheet for conversion of the filter cake to stainless steel slabs, as shown in Figure 1, has been used to develop the major equipment list given in Table 6. The facility has been identified by the following processing areas:

(a) Pelletizing and Sintering
(b) Smelting
(c) Scrap Melting and Holding
(d) AOD Refining
(e) Slab Casting

With details given in the Appendix, the total capital cost is $250 million.

8.3 Operating Costs

Unit costs for materials and services have been developed from in-house sources, other than the electric power cost of 2.2¢/kWh which was supplied by Canmet and the cost of the chromite concentrates which is a part of the study. Since the process concept entails producing 150,000 NTPA of 300 series and 50,000 NTPA of 400 series stainless, the average steel slab might analyse 16.5 percent chromium and 6.0 percent nickel. Therefore the average operating cost is shown in Table 7 to be $598.62 per net ton of average steel slab produced. This value is approximately $454 (US) per net ton of slab. As shown on page 28 of Appendix I, the equivalent average cost for the United States stainless producers is between $650 (US) and $750 (US) per net ton of slabs. Assuming a freight cost of $60.00 per ton to the present day major user market (Pittsburgh) the c.i.f. cost is $658.62 ($500.00 US).

Process optimization may result in cost savings which will serve to widen the $150 US to $250 US per net ton difference in operating costs between stainless slab from Manitoba and that currently produced in the United States. Some examples of areas where savings may be incurred would include plasma smelting and scrap preheating. However at present it is evident that nickel amounts to 32 percent, and the chromite concentrates to 14 percent of the total operating cost. Economies made with these two commodities will significantly impact on the total cost per ton of slab.

For example, operating costs have been developed utilizing electrolytic nickel at $2.00 per lb. The processing of nickel concentrates should result in cost saving.

TABLE 6
MAJOR EQUIPMENT LIST

(1) Drum Filter (12 ft. x 16 ft.)
(2) Disc Pelletizer (30 TPH)
(3) Sintering Furnace
(4) Raw Material Batching Plant
(5) 48 MVA Submerged Arc Furnace
(6) Venturi Scrubber
(7) Gas Holder
(8) 80 ton capacity Hot Metal Crane
(9) 2 x 75 ton Induction Furnaces
(10) 120 ton capacity Hot Metal Crane
(11) 75 ton AOD Installation c/w Baghouse
(12) 120 ton capacity Caster Crane
(13) 75 NTPH Slab Caster

TABLE 7

OPERATING COST TO PRODUCE 200,000 NTPA OF STAINLESS SLABS

	Components Unit Cost	Components Annual Cost	$/NT Slab (Avg)	
Chromite Concentrates	$63.08	$12,111,360	$ 60.57	
Electric Power	2.2¢/Kwh	$ 8,844,484	44.22	
Bentonite	$140/NT	$ 280,000	1.40	
Coke Fines	$ 80/NT	$ 320,000	1.60	
Lump Coke	$170/NT	$ 7,990,000	39.95	
Quartz	$ 30/NT	$ 900,000	4.50	
Dolomite/Limestone	$ 30/NT	$ 1,980,000	9.90	
Electrode Paste	$800/NT	$ 720,000	3.60	
Steel Scrap (Plain C)	$100/NT	$11,400,000	57.00	
Electrolytic Nickel	$2.0/pound	$45,000,000	225.00	
Oxygen	$0.15/100SCF	$ 1,548,750	2.13	
Nitrogen	$015/100SCF	$ 24,600	0.12	
Argon	$1.75/100SCF	$ 1,548,750	7.74	
Lime	$0.10/pound	$ 1,880,000	9.40	
Fluorspar	$0.25/pound	$ 550,000	2.75	
FeSi75	$0.35/pound	$ 1,680,000	8.40	
SiMn	$0.17/pound	$ 1,224,000	6.12	
Labor	(See P. 23)	$ 8,593,200	42.97	
Maintenance Supplies	(See P. 24)		37.50	
Misc. Supplies & Service	(See P. 24)		25.00	
Office/Overhead	(See P. 25)	$ 1,750,000	8.75	$598.62
Freight to Pittsburgh	(Class Rate)		60.00	
		TOTAL		**$658.62**

9.0 PRELIMINARY CASH FLOW CONSIDERATIONS

It was stated in the commentary, Section 4, that the grade of concentrate nor its cost are competitive in the North American market.

The preliminary capital cost for an integrated operation, mine through to "semi-finished" steel production, has been estimated to be in the range of $250M - $300M.

Therefore, the following computer readouts and graphs have been prepared to show the Internal Rate of Return, and the I.R.R. versus project life for the estimated range of capital costs and the operating costs which have been developed.

Since the major capital expenditure, the smelter, would command a minimum depreciated life of 20 years, the ore would have to be multiple sourced if the minimum mining cost were to be maintained through that period.

The results are encouraging and suggest that a detailed sensitivity analysis is warranted. For example, the production of 300 series steels only would substantially enhance the economics of the project.

BIRD RIVER - CAPEX $ 297 M
****

A. OPERATING DATA

1. MINING RATE

MTPY	1,225,000
STPY	1,350,318
STPD	3,700

2. MINE OPERATING COSTS TO 50 METRES

MINE UNIT COST PER MT ORE	6.22
MINE UNIT COST PER ST ORE	5.64
MINE UNIT COST PER ST CON	40.31
CHROME RATIO	0.14

3. MILL OPERATING COSTS

MILL UNIT COST PER ST ORE	3.19
MILL UNIT COST PER ST CON	22.79

4. SMELTER OPERATING COSTS

SMELT UNIT COST PER ST SL	538.05

5. TOTAL OPERATING COSTS

TOTAL UNIT COST PER ST SL	598.62

B. CAPITAL COSTS

MINE CAPEX K DLRS	32,000
MILL CAPEX K DLRS	15,276
SMELT CAPEX K DLRS	250,000
TOTAL CAPEX K DLRS	297,276

BIRD RIVER - CAPEX $ 297 M
****

C. SALES DATA

TONS SS SLAB	200,000
SERIES 300 RATIO	0.75
SERIES 400 RATIO	0.25
AVE SP 300	800.00
AVE SP 400	375.00
AVE SELL PRICE US	693.75
AVE SELL PRICE CAN	925.00
AVERAGE UNIT FREIGHT COST	60.00
AVE SP FOB PLANT	855.00

D. FINANCIAL

	1	2	10	15	20
REVENUE K DLRS	173,000	173,000	173,000	173,000	173,000
COST OF GOODS SOLD	119,723	119,723	119,723	119,723	119,723
GROSS MARGIN	53,277	53,277	53,277	53,277	53,277
INTERNAL RATE OF RETURN			0.12	0.16	0.17

1. CASH FLOW (EXCLUDING TAXES)

GROSS MARGIN	53,277	53,277	53,277	53,277	53,277
INTEREST	15,471	15,157	11,367	7,244	867
PRINCIPAL	3,444	3,757	7,547	11,671	18,048
CASH FLOW	34,362	34,362	34,362	34,362	34,362
IRROSH FLOW			0.03	0.08	0.10

DEBT RATIO	0.60
EQUITY RATIO	0.40
INTEREST RATE	0.0875

3. PRESENT VALUE

PV25	533,280
PV50	396,567
PV75	245,851
PV100	156,297
PV125	88,518
PV150	36,199
PV175	-4,938
PV200	-37,842
PV225	-64,581
PV250	-86,627
PV275	-105,046
PV300	-120,622
PV325	-133,938
PV350	-145,434
PV375	-155,449
PV400	-164,244

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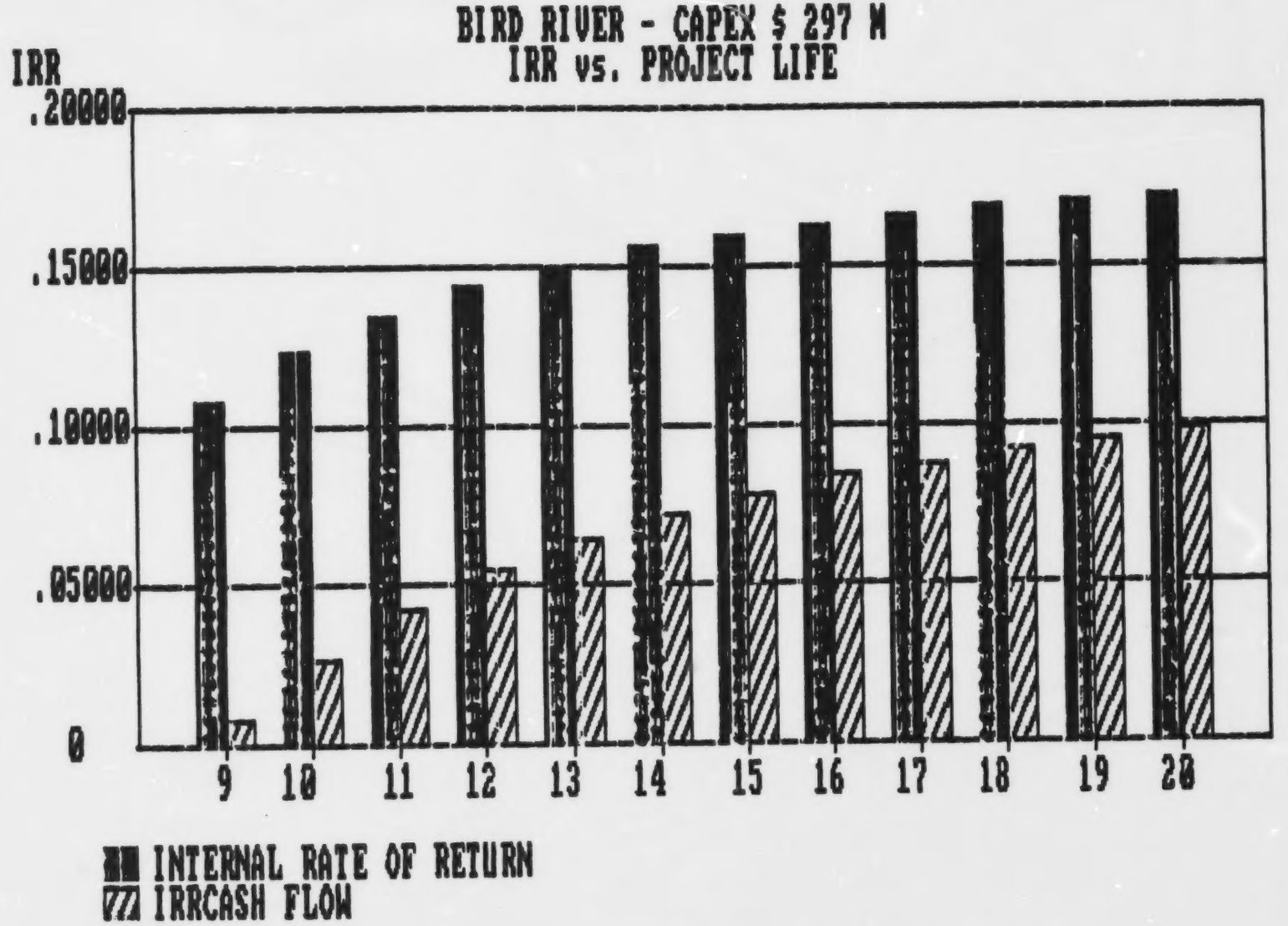
BIRD RIVER - CAPEX $ 297 M
IRR vs. PROJECT LIFE
IRR
.20000
.15000
.10000
.05000
0
9
10
11
12
13
14
15
16
17
18
19
20
INTERNAL RATE OF RETURN
IRRCASH FLOW

BIRD RIVER - CAPEX $ 250 M
****

A. OPERATING DATA

1. MINING RATE

MTPY	1,225.000
STPY	1,350.318
STPD	3,700

2. MINE OPERATING COSTS TO 50 METRES

MINE UNIT COST PER MT ORE	6.22
MINE UNIT COST PER ST ORE	5.64
MINE UNIT COST PER ST CON	40.31
CHROME RATIO	0.14

3. MILL OPERATING COSTS

MILL UNIT COST PER ST ORE	3.19
MILL UNIT COST PER ST CON	22.79

4. SMELTER OPERATING COSTS

SMELt UNIT COST PER ST SL	538.05

5. TOTAL OPERATING COSTS

TOTAL UNIT COST PER ST SL	598.62

B. CAPITAL COSTS

MINE CAPEX K DLRS	26,911
MILL CAPEX K DLRS	12,847
SMELT CAPEX K DLRS	210,242
TOTAL CAPEX K DLRS	250,000

BIRD RIVER - CAPEX $ 250 M
****

C. SALES DATA

TONS SS SLAB	200,000
SERIES 300 RATIO	0.75
SERIES 400 RATIO	0.25
AVE SP 300	800.00
AVE SP 400	375.00
AVE SELL PRICE US	693.75
AVE SELL PRICE CAN	925.00
AVERAGE UNIT FREIGHT COST	60.00
AVE SP FOB PLANT	865.00

D. FINANCIAL

	1	2	10	15	20
REVENUE K DLRS	173,000	173,000	173,000	173,000	173,000
COST OF GOODS SOLD	119,723	119,723	119,723	119,723	119,723
GROSS MARGIN	53,277	53,277	53,277	53,277	53,277
INTERNAL RATE OF RETURN			0.17	0.20	0.21

1. CASH FLOW (EXCLUDING TAXES)
.....

GROSS MARGIN	53,277	53,277	53,277	53,277	53,277
INTEREST	13,011	12,747	9,560	6,092	729
PRINCIPAL	2,896	3,160	6,347	9,815	15,178
CASH FLOW	37,370	37,370	37,370	37,370	37,370
IRRCASH FLOW			0.08	0.12	0.14

DEBT RATIO	0.60
EQUITY RATIO	0.40
INTEREST RATE	0.0875

2. PRESENT VALUE

PV25	580,536
PV50	413,943
PV75	293,127
PV100	203,573
PV125	135,794
PV150	83,475
PV175	42,338
PV200	9,434
PV225	-17,305
PV250	-39,351
PV275	-57,770
PV300	-73,346
PV325	-86,662
PV350	-98,158
PV375	-108,173
PV400	-116,968

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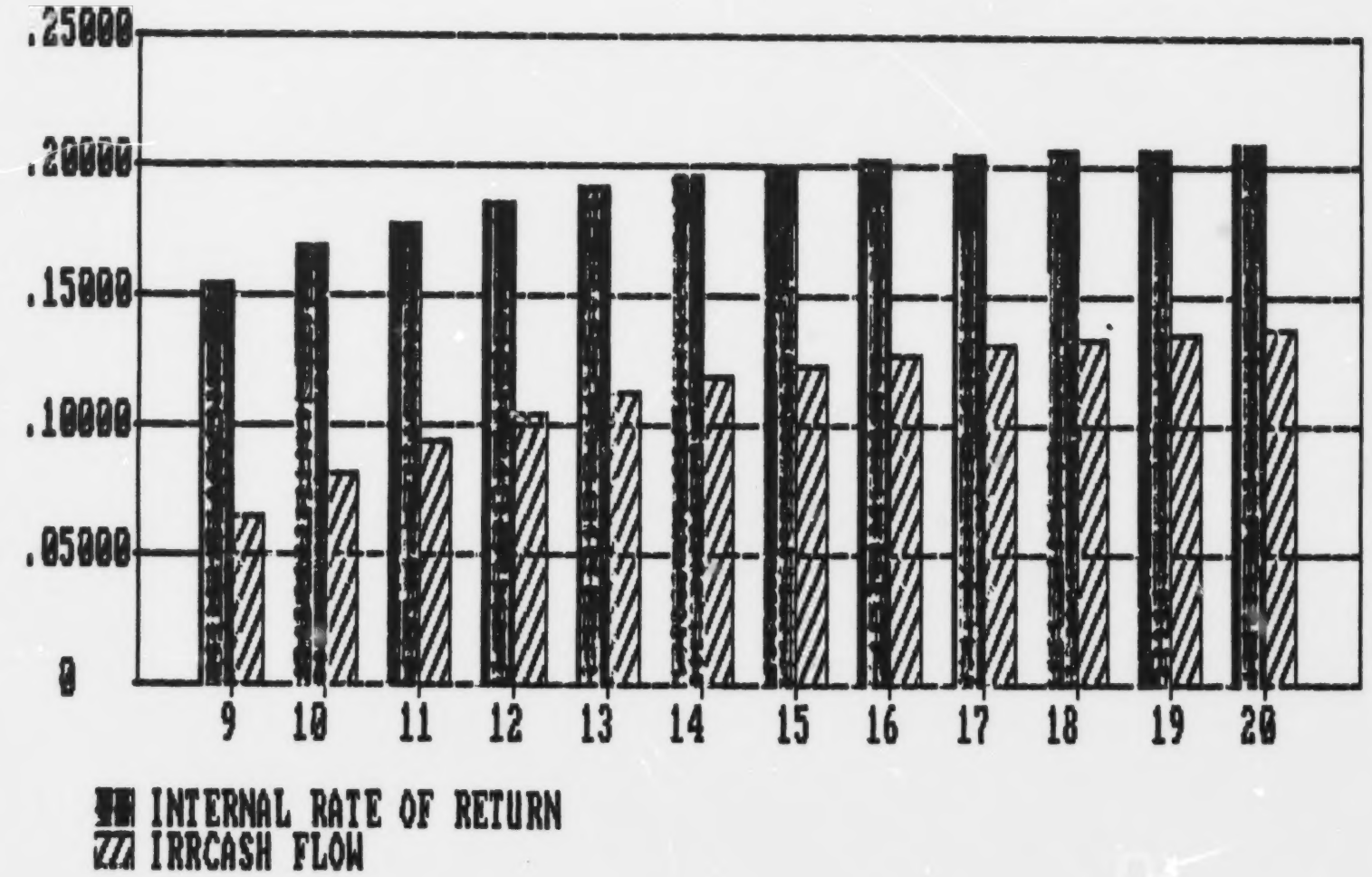
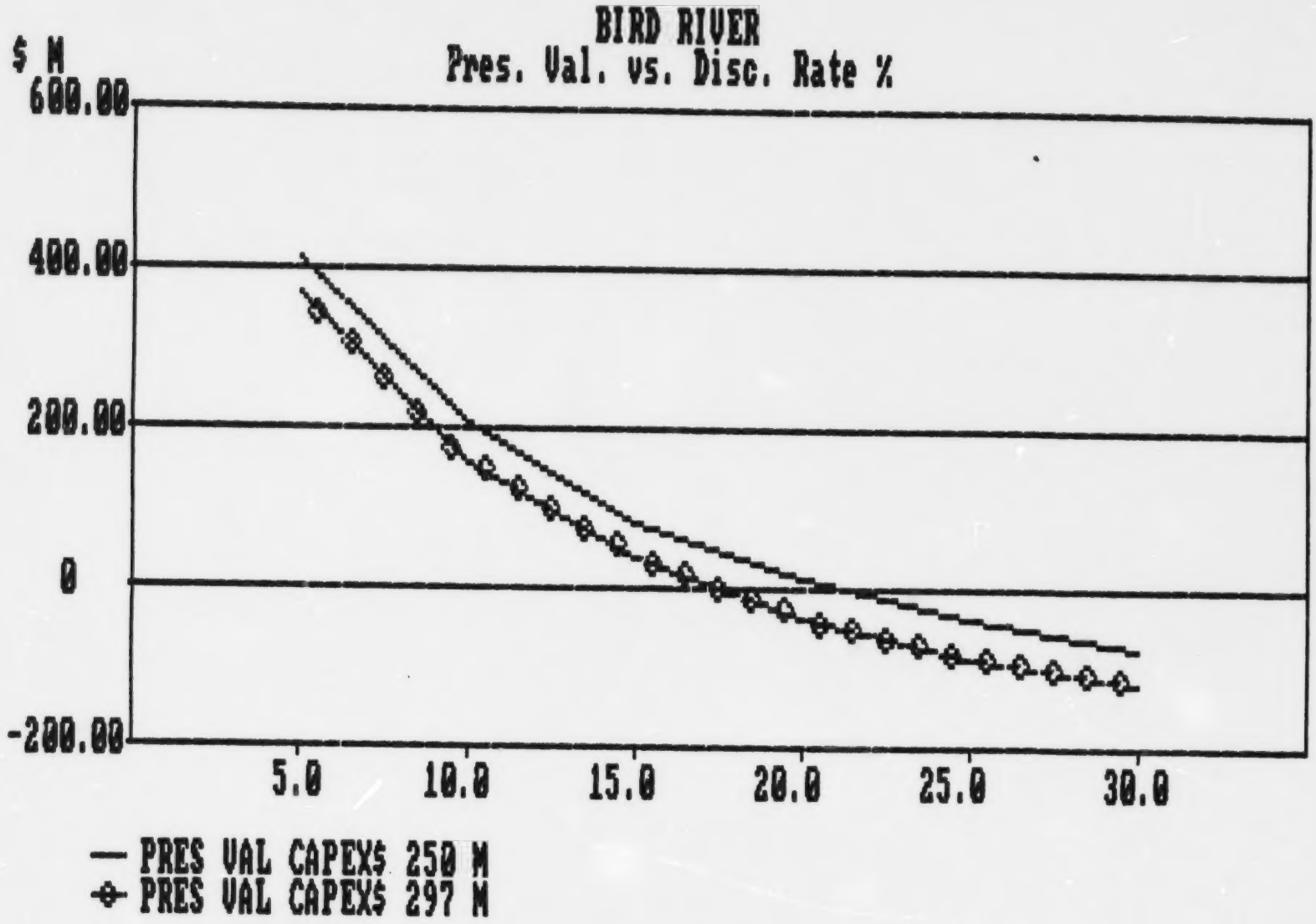
BIRD RIVER - CAPEX $ 250 M
IRR vs. PROJECT LIFE




BIRD RIVER
Pres. Val. vs. Disc. Rate %
$ M
600.00
400.00
200.00
0
-200.00
5.0
10.0
15.0
20.0
25.0
30.0
PRES VAL CAPEX$ 250 M
PRES VAL CAPEX$ 297 M

Appendix 1

REFERENCES

J.D.Bateman — 1943 CIM
Bird River Chromite Deposits, Manitoba

I.T.Hosain — 1978 Manitoba Mineral Resources Division
Summary and Evaluation of the Geophysical Data from the Open Assessment Files of the Bird River Greenstone Belt

B.B.Bannatyne and D.L.Trueman — Manitoba Energy and Mines 1982/86
Chromite Reserves and Geology of the Bird River Sill, Manitoba

D.M.Watson — Manitoba Energy and Mines 1985
Platinum - Palladium Distribution in Ultramafic Rocks of the Bird River Complex, Southeastern Manitoba

Lakefield Research — December 1986
Mineral Processing Investigations of Bird River Chromite Ore

Various Maps and Memoranda from Manitoba Energy and Mines

Appendix II

WORK SHEETS

CLASS I

INITIAL CAPITAL COST SUMMARY- MINING

	Service Life	Unit Cost $(000)	No. Req'd	Total Initial Cost
DRILLING				
RB Rotary Drill 9 7/8"	20	900	1	900
DT Hole Drill	10	600	1	600
BLASTING				
ANFO Truck	10	80	1	80
Powder Truck	10	15	1	15
LOADING				
Shovel 12 cy electric hydraulic	20	2200	1	2,200
2 - 988	5	800	2	1,600
HAULING				
Truck 50 Ton	8	800	10	700
AUXILLARY				
Water Truck	5-10	350	1	350
D-9 Dozer	5	450	1	450
D-8 Dozer	5	425	1	425
Grader	5	300	1	300
9 cu y F.E. Loader	5	600	1	600
ANCILLARY				
Lube/fuel Truck	5	90	1	90
Welding Truck	5	60	1	60
Repair Truck	5	60	1	60
Mechanic Truck	5	40	1	40
Cable Reel Truck	5	150	1	150
Backhoe	5	200	1	200
TOTAL				16,120

CLASS II

CAPITAL COSTS RE REPAIR SHOP AND AUXILLARY EQUIPMENT - MINING

		Unit/Cost	
3	5 Ton Crane	35	105
2	Sub Stations		300
1	5 Ton Fork Lift	150	150
2	Utility Trucks	30	60
1	Tractor Lowboy	120	120
1	Tire Forklift	120	120
1	Shop Forklift	60	60
20	Cable Spools	10	200
20	Cable Couplers	1	20
1	Switche, etc.	100	100
6	Light Plants	10	60
1	Aggregrate Crusher	800	800
5	Pickups	15	75
2	Man Vans	20	40
1	Fire Truck	60	60
1	Ambulance	50	50
1	Radio Communication System	200	200
20	Panier Poles and Aerial Cables	15	300
1	Engineering Inst./Misc.	50	50
1	Miscellaneous	170	170
		TOTAL	$3,040

MINE PLANT/PRIMARY CRUSHER

The mine plant facilities includes a combination building 150' x 50' x 40.

The one building covers the mine office, dry, warehouse and 3 -50' repair bays.

The estimated cost for the above is - $ 4,500,000

PRIMARY CRUSHER PLANT

Primary Crusher/Screen/Conveyor/Dust collection and cover is estimated at - $ 3,000,000

Miscellaneous - Pumphouse/Electrical - $ 2,000,000

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF 4

DESCRIPTION: SELECTION OF MINING RATE.

PROJECT NO. 9075

MADE BY ARP DATE 11 March 87

CHECKED BY DATE

TAYLOR'S RULE: $\frac{5(\text{expected tonnage})^{.75}}{\text{OPERATING DAYS.}} =$

ARP' RULE: $\frac{10\% \text{ of expected tonnage}}{\text{OPERATING DAYS.}}$

EXPECTED TONNAGE "ORE"

(a) 50m pit depth 5,372,500

(b) 100m pit depth 10745000

then

TAYLOR RULE = $\frac{5(10745000)^{.75}}{350}$ = 2682 TPD (938,700 T/Y)

ARP = $\frac{0.10(10745000)}{350}$ 3000 TPD (1,050,000 T/Y)

SELECTED MINING RATE 1,225,000 T/Y

3500 TPD

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 1 OF 4

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
ORE AND WASTE QUANTITIES	9075	DATE 11/3/87	DATE

I AREA stripped is calculated in the following way, where:

d = DEPTH of P.I.T
γ = ANGLE of SLOPE ; 45°
l = LENGTH of ORE BODY
DEPTH of OVERBURDEN = 3m
INCLINATION of RAMP 1:10
OVER BURDEN DENSITY: 2.0; ROCK = 2.8; ORE = 3.07

AREA over ORE $l \times w$
ALONG ORE $2 \times d \times$ Tan. of angle $90-\gamma$
ENDS $w \times 2d \times$ Tan of angle $90-\gamma$
Corners $\pi \times d^2 \tan^2 90-\gamma$
area over ramp $20 \times 10 \times d$

II WASTE ROCK REMOVAL.

CROSS SECTION of ORE BODY SHOWS THAT THE WASTE ROCK HAS THE SAME AREA AS A TRIANGLE WITH THE BASE $2 \times$ TAN of $90-\gamma \times d$ and HEIGHT d WHICH EQUAL.

(1) $\frac{1}{2}[2 \times d \times \text{TAN } 90-\gamma] \times d$ where γ = SLOPE AND d = DEPTH
VOLUME = AREA × LENGTH × S.G.

(2) ENDS; SAME AS I × WIDTH × S.G

3 FOUR CORNERS - VOLUME of CONE with $h = d$, RADIUS d × Tan of $90-\gamma$ · WHICH.
$d^3 \times \tan^2(90-\gamma) \times \frac{\pi}{3}$

4 OVER RAMP $10 d^2 \times \frac{20}{2}$.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 2 OF 4

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE
	9075	ARP / March 87	

WASTE REMOVAL.

ASSUME ORE is 14m AND ASSOCIATED WASTE FOR 28 m MINING FRONT = 14
TOTAL. ORE + WASTE = 28m.

(1) PIT DEPTH: 50m

AMOUNT of WASTE

PIT SLOPE ANGLE 55° d = 50m / 100m.

ALONG THE ORE	$\left[\frac{2 \times 50 \times .7}{2}\right] \times 50 \times 2500 \times 2.8$	12,250,000
AT THE ENDS	$\left[\frac{2 \times 50 \times .7}{2}\right] \times 28 \times 2500 \times 2.8$	6,860,000
AT THE CORNERS	$\left[50 \left[\frac{50 \times 50 \times .49}{3}\right] \times 3.14 \times 2.8\right.$	179,594
FIR THE RAMP	$10 \times 50 \times 50 \times \frac{20}{2} \times 2.8$	700 000
SUB TOTAL		19,989,594
Internal Waste	$14 \times 50 \times 2500 \times 2.8$	4900000
Grand Total		24,889,594
ORE	$14 \times 50 \times 2500 \times 3.07$	5372500
		30,262,094

S/R $\frac{\text{waste + ore}}{\text{ore}}$ $\frac{30.3}{5.4}$ = **5.61**

(2) PIT DEPTH – 100m

Along ORE.		
(a) ENDS	$\frac{2 \times 100 \times 70}{2} \times 100 \times 2500 \times 2.8$	49,000,000
· CORNERS	$70 \times 28 \times 2500 \times 2.8$	13,720,000
RAMP	$100 \times 100 \times 100 \times .49 \times \frac{3.14}{3} \times 2.8$	1,436,026
	$10 \times 100 \times 100 \times \frac{20}{2} \times 2.8$	2800000
SUB TOTAL.		67,006,026
INTERNAL WASTE	$14 \times 100 \times 2500 \times 2.8$	9800,000
		76,806,026

ORE : · 10,745,000 S/R **8.15**

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 3 OF 4

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
OVER BURDEN STRIPPING	9075	ARP DATE March 4/87	DATE

The ORE BODY IS APPROXIMATED AS A SLICE WITH A LENGTH of 2500m, A WIDTH of 14 m & PIT DEPTHS of 50 & 100 m.

FOR ALL PIT DEPTHS THE SLOPE OF OVER BURDEN HAS BEEN APPROXIMATED @ 45° .79 RAD.

SG of over burden ASSUMED @ 2 AND AVERAGE THICKNESS of 3 m

THE AMOUNT of OVER BURDEN IS ESTIMED @ 715000 TONNES, AS FALLOWS:

		TONNES
over the ore	2500 x 14 x 3. x 2 =	210,000
along the ore	2 x 2500 x 14 x 3 x 2 =	420000
at the ends	2 x 50 x 14 x 3 x 2 =	8400
at the corners	3.14 x 50 x 50 x 3 x 2 =	47100
over pit ramp	20 x 2500 x 3 x 2 =	30000
		715,000

HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

PAGE 1 OF 7

PROJECT BIRD RIVER CHROMITE

DATE FEB. 24/87

SECTION

BY

PROJ. Nº 9075

SUMMARY OF CONCENTRATOR CAPITAL COST ESTIMATE

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST HP	MAT.	LAB.	TOTAL
1	SAG MILLING CIRCUIT				1680			1,796,000
2	HEAVY MEDIA SEPARATION CIRCUIT				210			910,000
3	ROD MILLING & BALL MILLING				1850			1,365,000
4	GRAVITY SEPARATION CIRCUITS				—			563,000
5	CONCENTRATE FILTRATION CIRCUIT				260			210,000
6	TAILINGS DISPOSAL				75			56,000
	TOTAL EQUIPMENT COST				4075			4,900,000
	DIRECT COST ESTIMATE:							
	FREIGHT TO SITE 5%		245	000				
	SITE PREPARATION 5%		245	000				
	CONCRETE & FOUNDATIONS 15%		735	000				
	BUILDING 35%	1	715	000				
	MECH. INSTALLATION 20%		980	000				
	ELECT INSTALLATION 15%		735	000				
	PIPING 10%		490	000				
	HEAT & VENTILATION 5%		245	000				
	INSTRUMENTATION 5%		245	000				5,635,000
	TOTAL INSTALLED COST							10,535,000
	INDIRECT COSTS:							
	E.P.C.M. 15-20% (USE 20%)	2	107	000				
	CONTRACTORS' TEMPORARY 10%	1	054	000				
	CONTRACTORS' MARK-UP 15%	1	580	000				4,741,000
	OVERALL BATTERY LIMITS COST							15,276,000
	NOTE: TONNAGE CHANGE FROM 5000 TPD TO 3760 TPD							
	$\left(\frac{3760}{5000}\right)^{0.6} \times \$15,276,000$							12,900,000

HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

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PROJECT BIRD RIVER CHROMITE

DATE FEB 24/87

SECTION SAG MILLING CIRCUIT

BY

PROJ. Nº 9075

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST HP	MAT.	LAB.	TOTAL
1	5000 TONS PER CALENDAR DAY WITHDRAWN FROM ORE STOCKPILE. MINE RUN ORE							
	ALLOW FOR 2 FEEDERS TO COLLECTION CONVEYOR TO SAG MILL FEED CONVEYOR							
	SELECT:							
	VIBRATING FEEDERS	2						40,000
	COLLECTION CONVEYOR PLUS FEED CONVEYOR ALLOW 50' LIFT + 200' RUN				30			136,000
	REF: ASH KAUSHAL							
	SAG MILL 22'X8' INCLUDING FEED SPOUT & DISCHARGE							1,050,000
	REF: JACK CURRIE, ALLIS CHALMERS							
	SET OF CHROME-MOLY LINERS							220,000
	1600 HP, 240 R.P.M. SYNC. MOTOR INCLUDING AIR CLUTCH				1600			250,000
	SCREENING CIRCUIT (1¼") 6'X16' LOW-HEAD POLYPROPYLENE DECK				20			50,000
	REF: BILL SPICER, SALA							
	PUMPING ALLOWANCE				30			50,000
	TOTAL EQUIPMENT COST				1680			1,796,000

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HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

PAGE 3 OF 7

PROJECT BIRD RIVER CHROMITE

DATE FEB 24/87

SECTION HEAVY MEDIA SEPARATION CIRCUIT

BY JHH

PROJ. Nº 9075

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST HP	MAT.	LAB.	TOTAL
2	FEED PREPARATION SCREENS							
	2 - ALLIS CHALMERS LOW-HEAD							
	30 MESH @ $80,000 EACH				10			160,000
	ONE COMPLETE TRI-FLOW							
	HEAVY MEDIA PLANT, INCLUDING							
	MEDIA SCREENS, DENSIFICATION,							
	MAGNETIC SEPARATION, ALL							
	PIPING & WIRING TO A							
	CENTRAL BOX.							
	INCLUDES PUMPS + SUMP PUMP.							
	WASHED PRODUCT SUITABLE							
	FOR ROD MILL FEED							
	REF: DAVID GREEN, ~~WEMCO~~							
	OUTOKUMPU				200			750,000
	TOTAL EQUIPMENT COST				210			910,000

HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

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PROJECT BIRD RIVER CHROMITE

DATE FEB 24/87

SECTION

BY

ROD MILL & REGRIND CIRCUIT BALL MILL

PROJ.Nº 9075

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST			
						MAT.	LAB.	TOTAL
3	FEED 2500 TPD							
	BOND WORK INDEX = 21							
	GRIND REQ'D - ½" TO -30 MESH							
	SELECT:							
	12½' X 18' ROD MILL							550,000
	Ni-HARD LINERS							120,000
	1200 HP, 240 R.P.M. SYNC							
	MOTOR, INCLUDING AIR CLUTCH				1200			190,000
	BALL MILL TO REGRIND MIDDLINGS FROM BOTH ROUGHER & CLEANER GRAVITY SEPARATION CIRCUIT. FINAL GRIND UNKNOWN.							
	ALLOW ½ X ROD MILL CIRCUIT				600			430,000
	CLASSIFICATION & PUMPING							
	ALLOWANCE				50			75,000
	TOTAL EQUIPMENT COST				1850			1,365,000

HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

PAGE 5 OF 7

PROJECT BIRD RIVER CHROMITE

DATE FEB 24/87

SECTION GRAVITY SEPARATION CIRCUITS

BY

PROJ. Nº 9075

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
4	FEED 2500 TPD (208 TPH) MINUS 35 MESH GRIND TO ROUGHERS						
	REFERENCE: THOMAS J. FERREE, MINERAL DEPOSITS LTD., GOLDEN CO. (303-279-8932						
	ASSUME 3 T.P.H. PER TWIN-START SPIRAL = 69.3 UNITS SPIRALS ARE SUPPLIED IN BANKS OF 8 TWIN-STARTS						
	SELECT: 8 BANKS @ $38,000/BANK AUSTRALIAN MANUFACTURE						304,000
	DUTY 10 % $30,400						
	FREIGHT TO WEST COAST CANADA 10% 30,400						
	$60,800						61,000
	FEED DISTRIBUTION SYSTEM $8,000						
	+ DUTY 800						
	+ FREIGHT 800						
	9,600						10,000
	TOTAL ROUGHER CIRCUIT EQUIPMENT						375,000
	CLEANER CIRCUIT TO TREAT ROUGHER MIDDLINGS AND RECROUND ROUGHER AND CLEANER MIDDLINGS, PERHAPS BY REICHART CONES IS UNKNOWN						
	∴ ALLOW 1/2 COST OF ROUGHER CIRCUIT						188,000
	TOTAL EQUIPMENT COST						563,000

HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

PAGE 6 OF 7

PROJECT BIRD RIVER CHROMITE

DATE FEB 24/87

SECTION

BY

PROJ. Nº 9075

CONCENTRATE FILTRATION

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST HP	MAT.	LAB.	TOTAL
5	FEED RATE:							
	5000 TPD X 14 % = 700 TPD							
	REFERENCE: DON FOREMAN, of DORADO CANADA INC.							
	ASSUME FILTRATION RATE OF 100 LB/FT²/HOUR							
	i.e. $\frac{700}{24}$ X 2000 = 58,333 LB/HOUR							
	FILTRATION AREA REQUIRED = 583 FT²							
	SELECT: 12'X16' DAVIS FILTER (ALLOW 5 CFM/FT² VACUUM)							
	FILTER $100,000							
	VACUUM PUMP 65,000							
	AUXILIARIES 15,000							
	$180,000				250			189,000
	ALLOWANCE FOR CONVEYOR TO FURTHER PROCESSING				10			30,000
	TOTAL EQUIPMENT COST				260			210,000

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HATCH ASSOCIATES

ESTIMATE DETAILS

ACC'T Nº

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PROJECT BIRD RIVER CHROMITE

DATE FEB 24/07

SECTION TAILINGS DISPOSAL

BY

PROJ. Nº 9075

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST HP	MAT.	LAB.	TOTAL
6	DRY TONS TO TAILINGS DISPOSAL							
	= 5000 TPD X 86% = 4300 TPD							
	= 180 T.P.H.							
	%WT TPH S.G. U.S.GPM							
WATER	60 270 1.0 1080							
SOLIDS	40 180 2.8 257							
PULP	100 450 (1.35) 1337							
	REFERENCE:							
	BEALE KAZAZIAN (519) 824-7750							
	C.A.C. S.R.L. PUMPS							
	TWO 10X8X21 SRL-C @ $8,400 EACH							16,400
	INCLUDING OVERHUNG BASE & GUARD							
	MOTOR 75 HP, 1750 RPM				75			
	V-BELT DRIVE @ $900 EACH							1,800
	(GOOD FOR 2200 US GPM @ 100' HEAD)							
	TAILINGS PIPE ALLOW 3000 FEET							
	OF 8" SCHEDULE PIPE							
	REFERENCE: JACK CHESELMAN, DUPONT							
	821-3300							
	ALLOW $9/FOOT TO INCLUDE FUSION				—			27,000
	SINCE NO REAGENTS ARE							
	INVOLVED TAILINGS DISPOSAL							
	SHOULD BE NO GREAT PROBLEM							
	ALLOWANCE FOR DAM BUILDING							
	(EQUIPMENT COST ONLY) TARPS ETC.							10,800
	TOTAL EQUIPMENT COST							56,000

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HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 1 OF 5

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE OPERATING COST ESTIMATE	9075	DATE MAR 6/87	DATE

(NOMINAL OPERATING RATE OF 3760 SHORT TONS PER CALENDAR DAY)

SUMMARY

ITEM	$/TON OF ORE
1. SUPERVISION AND LABOUR	0.92
2. MAINTENANCE LABOUR AND SUPPLIES	0.33
3. LABORATORY SUPPLIES	0.11
4. GRINDING MILL LINERS	0.23
5. SCREEN CLOTH	0.08
6. GRINDING MEDIA	0.55
7. POWER	0.44
TOTAL	2.66
CONTINGENCY 20%	0.53
OPERATING COST ESTIMATE	$3.19

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 2 OF 5

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE OPERATING COST ESTIMATE	9075	DATE	DATE

(NOMINAL MILLING RATE OF 3,760 SHORT TONS PER CALENDAR DAY)

ITEM:

1. SUPERVISION & LABOUR		$/YEAR
MILL SUPERINTENDENT		55 000
ASS'T "		40 000
METALLURGIST		35 000
GENERAL FOREMAN		35 000
INSTRUMENT TECHNICIAN		32 000
SHIFT FOREMAN - 4 @ $30,000		120 000
CHIEF ASSAYER		30 000
		347 000
FRINGE BENEFITS @ 40%		139 000
TOTAL SALARIED PERSONNEL		486,000
16 OPERATORS @ $25,000	400,000	
2 DAY SHIFT LABOURERS	40,000	
3 LABORATORY HELPERS	75,000	
	515,000	
LABOUR BURDEN @ 40%	206,000	
OVERTIME & SHIFT PREMIUM 10%	52,000	
	773,000	
TOTAL HOURLY PAID PERSONNEL		773,000
TOTAL SUPERVISION & LABOUR		$1,259,000

COST PER TON OF ORE MILLED:

$$\frac{\$1,259,000}{3760 \times 365} = \$0.92$$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 3 OF 5

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE	9075	DATE MAR. 6/87	DATE

ITEM:

2. MAINTENANCE LABOUR & SUPPLIES

ESTIMATED AT 3.5% OF CAPITAL COST OF INSTALLED PLANT, INCLUDING BUILDING

$\$ 12,900,000 \times 0.035 = \$ 451,500$

$$\text{COST PER TON MILLED} = \frac{\$ 451,500}{3760 \times 365} = \$ 0.33$$

3. LABORATORY SUPPLIES:

ESTIMATED @ 30% OF LABORATORY PERSONNEL COST

ASSAYER + FRINGE BENEFITS	$= \$ 42,000$
3 HELPERS + LABOUR BURDEN	$= 112,500$
	$\$ 154,500$

$$\text{COST PER TON MILLED} = \frac{154,500}{3760 \times 365} = \$ 0.11$$

4. GRINDING MILL LINERS:

ASSUME 1 YEAR LIFE ON SAG MILL LINERS	$\$ 220,000$
" 2 " " " ROD " "	60,000
" 2 " " " BALL " "	30,000
	$\$ 310,000$

$$\text{COST PER TON MILLED} = \frac{\$ 310,000}{3760 \times 365} = \$ 0.23$$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF 5

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE	9075	DATE MAR 6/87	DATE

ITEM:

5. SCREEN CLOTH

PRIMARY GRINDING CIRCUIT (SAG MILLING)

ALLOW 2 CHANGES PER YEAR	$40,000
SAME TONNAGE TO 30 MESH SCREEN	40,000
ROD MILL SCREEN, ALLOW	20,000
BALL MILL SCREEN, ALLOW	10,000
	$110,000

$$\text{COST PER TON MILLED} = \frac{\$110,000}{3760 \times 365} = \$0.08$$

6. GRINDING MEDIA

SAG MILL ALLOW 0.5 LB/TON X 3760 TPD = 1880
ROD MILL ALLOW 1.5 LB./TON X 1880 TPD = 2820
BALL MILL ALLOW 2.0 LB/TON X 940 TPD = 1880
4580 LB./DAY

ASSUME GRINDING MEDIA DELIVERED COST = $450/TON

$$\text{COST PER TON MILLED} = \frac{4580 \times \$450}{2000 \times 3760} = \$0.55$$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 5 OF 5

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE	9075	DATE MAR 6/87	DATE

7. POWER COST

HORSEPOWER FROM CAPITAL ESTIMATE WORKSHEETS = 4075
= 3046 Kw

FACTOR ON TONNAGE RATE TO THE 0.6 POWER

$\left(\frac{3760}{5000}\right)^{0.6} \times 3046 = 2567$ Kw

ADDITIONAL POWER FOR UNESTIMATED ITEMS; LIGHTING, HEATING & VENTILLATION, UTILITIES ETC., ALLOW 20%

$KwH \text{ PER TON MILLED} = \frac{2567 \times 1.2}{156 TPH} = 20$

COST PER TON MILLED = $20 \times \$0.022 = \$0.44$

APPENDIX I

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 1 OF 31

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
BIRD RIVER CHROMITE — PROCESS CALCULATIONS.	9075	DPO'S DATE 3/87	DATE

PRODUCT : 200,000 NTPA Stainless Slabs, of which
150,000 NTPA would be 304 (18%Cr, 8%Ni)
and 50,000 NTPA would be 409 (12%Cr).

Cr in product = (150,000 × 0.18) + (50,000 × 0.12)
= 33,000 NTPA.

Recovery of Cr from slurry to slab = 84%.
=> Cr required in concentrates = $33,000 \times \frac{1}{0.84}$
= 39,300 NTPA Cr

Concentrates analyse 30% Cr_2O_3 with Cr:Fe ratio of 0.64:1
=> $39,300 \times \frac{152}{104} \times \frac{1}{0.30}$ = 191,462 NTPA Concentrates.

Say 192,000 NTPA Concentrates
125.8 NTPH.

MIXER: Add 1% Bentonite and 2% Coke Fines.
and concentrates contain 8% H_2O.
Therefore mix material consists of 192,000 NTPA Concs
1,920 NTPA Bentonite
3,840 NTPA Coke Fines
15,360 NTPA H_2O
213,120 NTPA Mix

85% Availability => $213,120 \frac{\text{tons}}{\text{yr}} \times \frac{1}{365 \times 24 \times 0.85} \frac{\text{yr}}{\text{hrs}}$

= 28.6 TPH.

PELLETIZER: 28.6 TPH + additional 7% H_2O
=> GREEN PELLETS = $28.6 \times \frac{1}{1-0.07}$ = 30.8 TPH.
Annual Green pellets = 30.8 × 365 × 24 × 0.85 ≈ 229,340 NTPA

Bentonite Feed Rate into Mixer:

$$\frac{192{,}000 \times 0.01}{365 \times 24 \times 0.85} \times 2000 = 516 \text{ pounds/hour}$$

Allow 1,000 pounds/hour max

2 week Bentonite storage = $(516 \times 24 \times 14)\frac{\text{pounds}}{2 \text{ wks}} \times \frac{1}{2000}$

= 86 NT, say 100 NT.

Coke Fines Feed Rate into Mixer:

$$\frac{192{,}000 \times 0.02}{365 \times 24 \times 0.85} \times 2000 = 1031 \text{ pounds/hour.}$$

Allow 1,500 pounds/hour max.

1 week Coke Fines storage = $(1031 \times 24 \times 14) \times \frac{1}{2000}$

= 86 NT, say 100 NT.

Water Additions.

7 % H_2O in the pelletizer.

$$\Rightarrow \left(\frac{192{,}000 \times 0.07 \times 2000}{365 \times 24 \times 0.85}\right)\frac{\text{pounds}}{\text{hr}} \times \frac{1}{62.4}\frac{\text{cuft}}{\text{pounds}} \times 7.48\frac{\text{USGal}}{\text{cuft}} \times \frac{1}{60}\frac{\text{hr}}{\text{mins}}$$

= 7.2 USGPM, say 10 USGPM Max.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 3 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Sintering Furnace receives 229,340 NTPA green pellets.

@ 15% H_2O and 2% Coke (equiv 1.7% C).

Weight loss during sintering = − 15% H_2O
− 1.5% Carbon (88% of C from coke)
−16.5%.

$\Rightarrow$ Fired Pellets = $229{,}340 \times (1 - 0.165) \cong 194{,}940$ NTPA
@ 85% availability

$\Rightarrow 194{,}940 \times \frac{1}{365 \times 24 \times 0.85} = 26.2$ NTPH.

Submerged-Arc Furnace: 97% Availability.
92% Cr Recovery
96% Fe Recovery.

1 NT Concentrates contain $2000 \times 0.30 \times \frac{104}{152} = 411$ pounds Cr.

and $411 \div 0.64 = 469$ pounds Fe

Therefore alloy contains: $411 \times 0.92 = 379$ pounds Cr
$469 \times 0.96 = 479$ pounds Fe
858 pounds Cr + Fe
+10% for C + Si = 86 pound C + Si
944 pounds Alloy

i.e 1 net ton concentrates will produce 944 pounds of alloy analysing 40.1% Cr
50.7% Fe
9.2% C + Si

Annual "Molten Fe Cr Alloy" Production = $192{,}000 \frac{\text{tons conc}}{\text{yr}} \times \frac{944}{2000} = 90{,}624$ NTPA.

& $\div (365 \times 24 \times 0.93) \Rightarrow 11.1$ NTPA Molten Fe Cr Alloy

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Assume slag : alloy ratio = 1.1 : 1

⇒ Annual slag production = 90,624 × 1.1 = 99,686 NTPA
& ÷ (365 × 24 × 0.93) ⇒ 12.2 NTPH.

Furnace Power : Ref Dressler, CIM, Vol 39, No. 893.

Power Consumption (KWh/tonne alloy) = 39[Cr] + 19[Fe] + 118[Si] + 590[SMR]

[Cr] = 40.1%; [Fe] = 50.7%; [Si] = 1.0%; [SMR] = 1.1 : 1

⇒ P.C = (39 × 40.1) + (19 × 50.7) + (118 × 1.0) + (590 × 1.1)
= 3294.2 KWh/t
× $\frac{2000}{2204}$ ⇒ 2989.3 KWh/net ton alloy.

Say 3,000 KWh/NT molten FeCr alloy.
Value consistent with Outokumpu & CMI data for pellets ✓.

∴ Furnace Power = 11.1 $\frac{\text{tons}}{\text{hr}}$ × 3,000 $\frac{\text{KWh}}{\text{ton}}$ = 33.3 MW.

Say 33 MW.

& @ 0.75 power factor ⇒ $\frac{33.3}{0.75}$ = 44.4 MVA transformer capacity.

Say 3 × 16 MVA single phase transformers for total of 48 MVA capacity.

Concentrates will analyse 30.0% Cr_2O_3
15.3% FeO (11.9% Fe)
17.0% Fe_3O_4 (11.9% Fe)
sub-total 62.3% (Cr + Fe)
$\Rightarrow$ 37.7% Al_2O_3, MgO, SiO_2
100.0% TOTAL.

Carbon Requirement:

(i) $Cr_2O_3 + 3C = 2Cr + 3CO$
Mol wt. 152 36 104

Annual Cr in molten FeCr Alloy = 90,620 × 0.401
$\Rightarrow$ C requirement for Cr Red^n = $90,620 \times 0.401 \times \frac{36}{104}$ = 12,578 NTPA C. *

(ii) $Fe_2O_3 + 3C = 2Fe + 3CO$
159.7 36 111.7

Annual Fe (from Fe_2O_3) in molten FeCr Alloy = 90,620 × 0.5 × 0.507
$\Rightarrow$ C requirement for Fe Red^n (from Fe_2O_3) = $90,620 \times 0.5 \times 0.507 \times \frac{36}{111.7}$
= 7,403 NTPA C. *

(iii) $FeO + C = Fe + CO$
71.85 12 55.85

Annual Fe (from FeO) in molten FeCr Alloy = 90,620 × 0.50 × 0.507
$\Rightarrow$ C requirement for Fe Red^n (from FeO) = $90,620 \times 0.5 \times 0.507 \times \frac{12}{55.85}$
= 4,936 NTPA C *

(iv) $SiO_2 + 2C = Si + 2CO$
60 24 28

Annual Si in molten FeCr alloy = 90,620 × 0.01
& C requirement = $90,620 \times 0.01 \times \frac{24}{28}$ = 777 NTPA C *

(v) C in molten FeCr alloy = 90,620 × 0.082
= 7,431 NTPA C *

Σ_1* = 33,125 NTPA C.

+20% for burn-off

⇒ 33,125 × 1.20 = 39,750 NTPA C.

@ 85% Carbon in coke, with 14% Ash.

⇒ Annual Coke Requirement = $39,750 \frac{\text{tons}}{\text{yr}} \times \frac{1}{0.85}$

= 46,764 NTPA Coke

Ash from coke to slag = 46,764 × 0.14
= 6,547 NTPA. *

Gangue from concs to slag = (total gangue $Al_2O_3 + MgO + SiO_2$) − ($\frac{60}{32}$ × 90,620 × 0.01 $SiO_2 \rightarrow Si$ in alloy)

= (90,620 × 0.377) − (1,942)

= 32,222 NTPA. *

TOTAL SLAG = 99,690 NTPA.
slag from * above = 38,769 NTPA
⇒ slag from fluxes = 60,921 NTPA.

Assume 50% of flux will be quartz
$\Rightarrow$ Quartz = 30,461 NTPA.

Assume other 50% will be dolomite/limestone blend.

$CaCO_3$ is 44% CO_2.
$MgCO_3$ is 52% CO_2
Assume Dol/L'stone Blend analyses 46% CO_2.

$$\Rightarrow 60{,}921 \times 0.50 \times \frac{1}{0.46} = 66{,}216 \text{ NTPA Dol/L'stone}$$

(NOTE: Cr + Fe as oxides in slag not included.)

Electrode Paste: @ 3 kg/MWh.

$$\Rightarrow 90{,}620 \frac{\text{tons alloy}}{\text{yr}} \times 3.0 \frac{\text{MWh}}{\text{ton alloy}} \times \left(\frac{3 \times 2{,}204}{2000}\right) \frac{\text{net tons}}{\text{MWh}}$$

$$= 899 \text{ net tons paste/year.}$$

Smelting Furnace Power = $2.0 \frac{\text{MWh}}{\text{net ton}} \times 90{,}620 \frac{\text{net tons}}{\text{yr}}$

$$= 271{,}860 \text{ MWh/yr.}$$

Submerged-Arc Furnace would be tapping every 2 hours
$\Rightarrow$ Ladle Capacity = 11.1 x 2 = 22.2 tons molten FeCr alloy.

Need TWO induction furnaces for scrap melting ahead of the AOD vessel, one for 400 series steel and the other for 300 series steel.

(A) 400 series induction furnace

22 tons molten FeCr alloy @ 40% Cr
+ plain carbon steel scrap to lower Cr in "transfer metal" to 12.5% Cr (allows for Cr loss during AOD blow).

$$\Rightarrow \quad 0.125 = \frac{22 \times 0.40}{(22 + y)}$$

Where y = net tons scrap to be added.

$$\Rightarrow \quad 0.125(22 + y) = 22 \times 0.40$$
$$\Rightarrow \quad y = 48.4 \text{ net tons scrap.}$$

$\therefore$ Induction Furnace capacity = 22.2 + 48.4
= 70.6 net tons.

Use a 75 net ton capacity furnace.

Max. power rating : 450 Kwh/ton scrap melted
1 hour melting time.

$$\Rightarrow 48.4 \frac{\text{net tons}}{\text{hr}} \times 0.450 \frac{\text{MWh}}{\text{ton}} = 21.8 \frac{\text{MWh}}{\text{hr}}$$

say 22 MW furnace power for melting.
Holding would be at some lower power level, say 6 MW.
and @ power factor of 0.75 $\Rightarrow \frac{22}{0.75} = 29.3$, say 30 MVA transf.

Therefore Induction Furnace Cycle would be
(A) charge molten FeCr alloy : ⇔
(B) add and melt scrap : 22 MW for 1 hour
(C) hold at temp : 6 MW for 1 hour.

⇒ power/cycle = 22 + 6 = 28 MWh

Since 25% of product is 400 series stainless,

⇒ (0.25 × 90,620) NTPA Molten FeCr Alloy used for 400 series @ 22.2 NT —— " —— /fce cycle.

⇒ $\frac{0.25 \times 90{,}620}{22.2}$ = 1020 furnace cycles/year.

or 2,040 operating hours/year.
(23% utilization factor)

& @ 28 MWh/fce cycle
⇒ 1020 × 28 = 28,560 MWh/yr for 400 series ind fce operation

Assume there is a second furnace sharing the common power supply in order to avoid Ni contamination from 300 series into 400 series steels.

(B) 300 series induction furnace.

33 tons molten FeCr alloy @ 40% Cr.
NOTE: tap smelting fce every 3 hours to fully utilize ind fce capacity.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 10 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Add plain carbon steel scrap to lower Cr in "transfer metal" to 20.5% Cr (allow for Cr loss during AOD blow as well as for Ni dilution in AOD)

$$\Rightarrow 0.205 = \frac{33 \times 0.40}{33 + x}$$

Where x = net tons scrap to be added.

$$\Rightarrow 0.205(33+x) = 33 \times 0.40$$

$$\Rightarrow x = 31.4 \text{ net tons scrap}$$

Induction Fce capacity = 33.3 + 31.4 = 64.7 net tons. ✓SK.

To melt the 31.4 net tons scrap in 1 hour, @ 450 kWh/ton

$$\Rightarrow 31.4 \times 0.450 = 14 \frac{\text{MWh}}{\text{hr}}$$

Furnace will be operated at 14 MW (64% of max. rating) for melting

$\Rightarrow$ 14 MWh/cycle for melting

2 hours holding @ 6 MW $\Rightarrow$ 12 MWh/cycle for holding

$\Rightarrow$ power/cycle = 26 MWh.

(90,620 × 0.75) NTPA molten FeCr alloy used for 300 series

@ 33.3 NTPA ——— " ——— /fce cycle

$$\Rightarrow \frac{90,620 \times 0.75}{33.3} = 2041 \text{ furnace cycles/yr.}$$

or 6122 operating hours/yr.
(70% utilization factor)

& @ 26 MWh/cycle

$\Rightarrow 2041 \times 26 = 53,066$ MWh/yr for 300 series incl fce operation

Check: power plant utilization = 23 + 70 = 93%.

Power Requirement for Induction Furnaces = 28,560 (400 series)
+ 53,066 (300 series)
Total = 81,626

Scrap Requirement = $48.4 \frac{\text{tons}}{\text{cycle}} \times 1020 \frac{\text{cycles}}{\text{yr}}$ = 49,368 tons (400 series)

+ 31.4×2041 = 64,087 tons (300 series)

Total = 113,455 tons/yr

(Note: have assumed 100% recovery from scrap → steel)

400 Series Transfer Metal:

Analysis:
(Scrap @ 0.5%C, 0.5%Si, 1.0% others, 98% Fe.)

Carbon level = $\frac{(48.4 \times 0.005) + (22.2 \times 0.002)}{70.6 (48.4 + 22.2)} \times 100 \cong 2.9\%C.$

Silicon level = $\frac{(48.4 \times 0.005) + (22.2 \times 0.01)}{(48.4 + 22.2)} \times 100 \cong 0.7\%Si$

Chromium level = $\left(\frac{22.2 \times 0.401}{48.4 + 22.2}\right) \times 100 = 12.6\%Cr.$

Iron level = $\frac{(22.2 \times 0.507) + (48.4 \times 0.98)}{(48.4 + 22.2)} \times 100 = 83.1\%Fe.$

tonnage: $(48.4 + 22.2)\ \frac{tons}{cycle} \times 1020\ \frac{cycles}{yr} = 72,012$ NTPA.

300 series Transfer Metal:

analysis:

Carbon level = $\frac{(31.4 \times 0.005) + (33.3 \times 0.082)}{(31.4 + 33.3)} \times 100 \cong 4.5\%C$

Silicon level = $\frac{(31.4 \times 0.005) + (33.3 \times 0.01)}{(31.4 + 33.3)} \times 100 \cong 0.8\%Si$

Chromium level = $\frac{(33.3 \times 0.401)}{(31.4 + 33.3)} \times 100 = 20.6\%Cr.$

Iron level = $\frac{(33.3 \times 0.507) + (31.4 \times 0.98)}{(31.4 + 33.3)} \times 100 = 73.7\%Fe.$

tonnage: $(31.4 + 33.3)\ \frac{tons}{cycle} \times 2041\ cycles/yr = 132,053$ NTPA.

Considering the fact that transfer metal will analyse between 4.5 and 2.9% Carbon, the AOD vessel will need to be equipped with a top lance used for rapid decarburization down to approx 1% without the use of inert gases (N_2 or Ar).

A 75 ton AOD vessel will be considered as the preferred capacity.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 13 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Decarburization to 1%C with O_2 Top Lance

Assume:
Top Lance capacity : 3,000 SCFM O_2 thru 6 inch lance.

	$2C$	+	O_2	=	$2CO$
moles	2		1		2
mol. wt	24		32		56

300 series Transfer Metal : 64.7 tons/cycle @ 4.5%C.
Toplance C Removal = $64.7 \times 2000 \times (0.045 - 0.010) = 4,529$ lbs C.
& ÷ 12 ⇒ 377 lb-moles C to be removed.
From above equation, 2 lb-moles C removed by 1 lb-mole O_2.
⇒ 377 ÷ 2 = 189 lb-moles O_2 required.
@ 379 SCF/lb-mole
⇒ $189 \times 379 = 71,520$ SCF O_2 required

@ 3,000 SCFM ⇒ $\frac{71,520}{3,000} = 23.8$ minutes of blowing thru top lance.

400 series Transfer Metal : 70.6 tons/cycle @ 2.9%C.
Top lance C removal = $70.6 \times 2000 (0.029 - 0.010) = 2,683$ lbs C.
⇒ 224 lb-moles C to be removed.
& ÷ 2 ⇒ 112 lb-moles O_2 required
× 379 ⇒ 42,367 SCF O_2 required

@ 3,000 SCFM ⇒ $\frac{42,367}{3,000} = 14.1$ minutes of blowing thru top lance.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 14 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

After initial Carbon reduction to 1% by top lance, the following sequence takes place :-

	Gas Ratio	Tuyere Utilization	Time Duration
1st Blow	3:1/O_2:N_2	100%	?
2nd Blow	1:1/O_2:Ar	100%	10 mins.
3rd Blow	1:3/O_2:Ar	75%	5 mins.
STIR	100% Ar	75%	5 mins.

@ Tuyere Capacity of 2000 SCF/hr. nation.

$\Rightarrow$ Total tuyere capacity $= 2000 \times 75 \times \frac{1}{60}$

$= 2500$ SCFM.

300 series: Total C after top lance $= \left(64.7 \times 0.045 - \frac{4529}{2000}\right) = .065$ tons C. (1294 pounds C)

Carbon end-point = 0.05% C Typical.

3rd Blow $O_2 = \left(\frac{2500 \times 5 \times 0.75 \times 0.25}{379}\right)$ lb-moles $O_2 \times 2 \frac{\text{lb moles C}}{\text{lb moles } O_2} \times 12 \frac{\text{lbs C}}{\text{lb mole C}}$

(2,344 SCF O_2)

$= 148$ lbs C removed.

2nd Blow $O_2 = \left(\frac{2500 \times 10 \times 1.00 \times 0.50}{379}\right) \times 2 \times 12 = 792$ lbs C removed

(12,500 SCF O_2)

C to be left = $\frac{0.08}{100} \times 64.7 \times 2000 = 104$ lbs C.

∴ to be removed during 1st Blow = 1294
− 104 (to be left)
− 148 (3rd Blow)
− 792 (2nd Blow)
250 pounds

1st. Blow = $\left(\frac{250}{12}\right)$ lb moles C $\times \frac{1}{2}$ $\frac{\text{lb mole } O_2}{\text{lb moles C}} \times 379$ $\frac{\text{SCF } O_2}{\text{lb mole } O_2}$

$= 3948$ SCF O_2.

O_2 Blow rate = $2500 \times 1.00 \times 0.75 = 1875$ SCFM

⇒ Blow Time = $3948 \div 1875 = 2.1$ minutes

Now total O_2 Consumed = 71,520 SCF (top lance) (0.13)
3,948 SCF (1st Blow)
12,500 SCF (2nd Blow)
2,344 SCF (3rd Blow)
90,312 SCF Total Theoretical

Add 15% for "losses" ⇒ 13,547 SCF
TOTAL Actual = 103,859 SCF O_2 / cycle

Corrected 1st Blow Duration for (3,948 + 13,547) = 17,495 SCF O_2

⇒ 17,495 SCF $O_2 \times \left(\frac{1}{2500 \times 1.00 \times 0.75}\right) \frac{\text{min}}{\text{SCF}_{O_2}} = 9.3$ mins.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 16 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

300 series transfer metal to 300 series blown metal.

Sequence:

TOP Lance	23.8 mins.	
1st Blow	9.3 mins	
2nd Blow	10.0 mins	
3rd Blow	5.0 mins	
STIR	5.0 mins.	
Additions/Tapping/Repair	30.0 mins	(Allowance)
Turnaround time	83.1 mins.	✓ OK.

Assume Nickel is added as electrolytic: 99.9 % Ni

(64.7 tons transfer metal) − (64.7 × (0.045 − 0.005) tons C removed)

= 62.11 tons carbon-free metal.

For 8 % Nickel in stainless, 99 % Recovery.

$$\Rightarrow \text{Nickel in stainless} = 0.08 = \frac{(0.99x)}{62.11 + x}$$

$$\Rightarrow 0.08(62.11 + x) = 0.99x$$

$$62.11 = 12.38x - x$$

$$\Rightarrow x = 5.5 \text{ tons Nickel required/cycle}$$

Therefore 300 series blown metal / steel weight = 62.1 + 5.5

= 67.6 Net tons/cycle.

@ 2041 cycles/yr ⇒ 137,972 NTPA 300 series Blown Metal

Nickel required = 5.5 × 2041 = 11,226 NTPA Ni

& × 2000 = 22.5 million pounds/yr Ni

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 17 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

300 series blown metal analysis: 67.6 net tons/cycle.

$$\text{Nickel} = \frac{5.5 \times 0.999 \times 0.99}{67.6} \times 100 = 8.0\%\,Ni$$

$$\text{Chromium} = \frac{33.3 \times 0.401 \times 0.92}{67.6} \times 100 = 18.2\%\,Cr$$

$$\text{Carbon} = \frac{104}{2000} \times \frac{1}{67.6} \times 100 = 0.08\%\,C.$$

Nitrogen consumption/cycle = $2{,}500 \times 9.3 \times 1.00 \times 0.25$

= 5,813 SCF N_2

& x 2041 cycles/yr ⇒ 11.9 MSCF/year. N_2

Argon consumption/cycle = $(2500 \times 10 \times 1.00 \times 0.50)$ 12,500

$+(2500 \times 5 \times 0.75 \times 0.75)$ 7,031

$+(2500 \times 5 \times 0.75 \times 1.00)$ 9,375

= 28,906 SCF/cycle Ar

@ 2041 cycles/yr ⇒ 59.0 MSCF/year. Ar

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 18 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

400 series: Total C after toplance = $\left(70.6 \times 0.029 - \frac{2,643}{2000}\right)$

= 0.71 tons C (1412 pounds C)

3rd Blow O_2 = 2,344 SCF O_2 & 148 lbs C removed } (see p14)
2nd Blow O_2 = 12,500 SCF O_2 & 792 lbs C removed }

C to be left = $\frac{0.15}{100} \times 70.6 \times 2000$ = 212 lbs C

∴ to be removed during 1st Blow = 1,412
− 212 (to be left)
− 148 (3rd Blow)
− 792 (2nd Blow)
260 pounds

1st Blow (see p.15) = $\frac{260}{12} \times \frac{1}{2} \times 379$ = 4,106 SCFM O_2

@ 1875 SCFM O_2 Blow Rate
⇒ Blow Time = $\frac{4106}{1875}$ = 22 minutes

Now total O_2 consumed = 42,367 SCF (toplance) (p.13)
4,106 SCF (1st Blow)
12,500 SCF (2nd Blow)
2,344 SCF (3rd Blow)
61,317 SCF Total Theoretical

Add 15% for "losses" ⇒ 9,198 SCF
TOTAL Actual = 70,515 SCF O_2/cycle.
ANNUAL O_2 @ 1020 cycles/yr ⇒ 71.9 MSCF O_2/yr.
Corrected 1st Blow Duration for (4,106 + 9,198) = 13,304 SCF O_2
⇒ $\left(13,304 \times \frac{1}{2500 \times 1.00 \times 0.75}\right)$ = 7.1 mins. (See p.15)

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 19 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

400 series transfer metal to 400 series blown metal

Sequence

TOP Lance	14.1 mins.
1st Blow	7.1 mins.
2nd Blow	10.0 mins
3rd Blow	5.0 mins
STIR	5.0 mins
Additions/Tapping/Repair	30.0 mins.
Turn-around time	71.2 mins. ✓ ok.

Carbon-free metal = (70.6 tons transf. metal) − (70.6 × (0.029 − 0.0015) tons C removed.)

= 70.6 − 1.94 = 68.7 net tons blown metal/cycle

@ 1020 cycles/yr ⇒ 68.7 × 1020 = 70,074 NTPA 400 series Blown metal

& Blown metal analysis: 68.7 net tons/cycle.

$$\text{Chromium} = \frac{22.2 \times 0.401 \times 0.92}{68.7} \times 100 = 11.9\%\text{Cr}.$$

$$\text{Carbon} = \frac{212}{2000} \times \frac{1}{68.7} \times 100 = 0.15\%\text{C}.$$

Nitrogen consumption/cycle = 2500 × 7.1 × 100 × 0.25

= 4,438 SCF N_2

& @ 1020 cycles/yr ⇒ 4.5 MSCF/year N_2

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 20 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Argon consumption / cycle = 28,906 SCF Ar. (see p. 17)

(i.e. 1020 cycles/yr ⇒ 29.5 MSCF/yr Ar.

GAS USAGE SUMMARY

	300 series		400 series		TOTAL
	SCF/Heat	MSCF/year	SCF/Heat	MSCF/year	MSCF/year.
O_2	103,859	212.0	70,515	71.9	283.9
N_2	5,813	11.9	4,438	4.5	16.4
Ar	28,906	59.0	28,906	29.5	88.5

FLUXES INTO AOD.

90 pounds lime / NT Blown metal.
10 pounds fluorspar / NT Blown metal.

Total Blown Metal = 138,000 + 70,000 = 208,000 NTPA

Annual Lime Requirement = $208,000 \times \frac{90}{2000}$ = 9,360 NTPA Lime

Annual Fluorspar Requirement = $208,000 \times \frac{10}{2000}$ = 1,040 NTPA Fluorspar.

ALLOYS INTO AOD.

FeSi75 : 45 pounds/NT Blown metal. (Avg. all grades)
SiMn : 35 pounds/NT Blown metal (Avg. all grades).

@ 208,000 NTPA Blown metal (all grades)

⇒ Annual FeSi75 Requirement = $208{,}000 \times \frac{45}{2000}$

= 4,680 NTPA

Annual SiMn Requirement = $208{,}000 \times \frac{35}{2000}$

= 3,640 NTPA.

Electric Power:

Smelting Furnace	271,860 MWh/yr. (p7)
Induction Furnaces	81,626 MWh/yr (p.11)
Sub-total	353,486 MWh/yr.
+20% for Auxiliaries	70,697 MWh/yr
Total	424,183 MWh/yr

SLAB CASTER: - 70 Net Tons Blown Metal
- 1 hour max casting time
Say 75 NTPH Casting Capacity.

$$\text{Operating Rate} = 200{,}000 \frac{\text{tons}}{\text{yr}} \times \frac{1}{75} \frac{\text{hr}}{\text{tons}} \times \frac{1}{365 \times 24} \frac{\text{yr}}{\text{hrs}} \times 100$$

$$= 30.4\%.$$

400 Series Blown Metal: 68.7 net tons/cycle
@ 75 NTPH
$$\Rightarrow \frac{68.7}{75} \times 60 = 55 \text{ mins/caster cycle}$$

300 Series Blown Metal: 67.6 net tons/cycle
@ 75 NTPH
$$\Rightarrow \frac{67.6}{75} \times 60 = 54 \text{ mins/caster cycle.}$$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 23 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

OPERATING LABOR. (Basis ~~40~~ hrs/wk, 4 shifts)

	shift	days	total
(A) Pelletizing & Sintering	4	1	17
(B) Smelting	9	1	37
(C) Scrap Melting & Holding	2	–	8
(D) AOD Refining	4	1	17
(E) Slab Casting	6	1	25
	25	4	104

MAINTENANCE LABOR.

@ Ratio 2 prod^n : 1 mtce.

	12	2	50
TOTAL LABOR	37	6	154

Straight time cost = $154 \times 40 \frac{\text{hrs}}{\text{wk}} \times 50 \frac{\text{wks}}{\text{yr}} \times 18 \frac{\$}{\text{hr}} = \$5,544,000$

$+ 0.35\times$ for fringes $\$1,940,400$

$+ 0.20\times$ for overtime allowance $\$1,108,800$

Total Labor Cost = $\$8,593,200$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 24 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Fuel Gas: Assume plant is fully self-sufficient — furnace gas is a total replacement for purchased natural gas (or even fuel oil).

(Note: Outokumpu with 50-70 MW of furnace power can export 50% of the gas to its steel plant which includes finishing lines — assumption OK.)

Water: Assume negligible cost effect.

Refractories:

Smelting furnace — 5 year life.
Induction furnaces —
40 ton ladles (for FeCr alloy) —
80 ton transfer ladles —
80 ton bottom pour ladles —
Tundish liners —

include with maintenance materials @ 3% of capital

⇒ Mtce. Supplies = 0.03 × $250 × 10^6/year.

= 7,500,000/year or $37.50/NT slabs

Misc supplies & services @ 2% of capital

= 0.02 × $250 × 10^6/year

= 5,000,000/year or $25.00/NT slabs.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 25 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

PLANT OFFICE / OVERHEAD.

Divisional Manager:	1
Production Superintendents	2
Asst Prod'n Superintendents	2
Mine Superintendents	2
Controller / Accountant	1
Personnel Officer & Assistant	2
Draftsmen / Drawing Control	2
Purchasing Agents	2
Stores Assistants	4
Chemist	1
Asst Chemist	1
Secretaries / Typists / Receptionist	6
Clerks / Print room / etc	5
	31

$1.25 x 10^6 / yr incl fringes.

Supplies, services, vehicles, etc. $0.50 x 10^6 / yr.

TOTAL = $1.75 x 10^6 / yr.

SCRAP / FeSi75 CORRECTION.

With the large amounts of decarburization taking place in the AOD, it is likely that a substantial amount of scrap can be added directly into the AOD as a coolant, as well as cutting the fuel supplied by the FeSi75.

300 series: 33.3 NT FeCr Alloy + 31.4 NT Scrap → 62.1 NT Blown Metal (Cordl. & 67.6 NT Blown Metal (w.N.

400 series: 22.2 NT FeCr Alloy + 48.4 NT Scrap → 68.7 NT Blown Metal

Say, 68 NT Blown Metal / AOD cycle.
@ 20% scrap addition, ⇒ 68 × 0.2 = 13.6 NT Scrap / AOD cycle

∴ Induction Furnace operation would become :-

for 300 series, 33.3 NT FeCr Alloy + (31.4 − 13.6) NT Scrap
The 17.8 NT Scrap is to be melted in 1 hour.
@ 450 Kwh/ton scrap ⇒ 17.8 × 0.45 = 8.0 MWh to melt.
∴ Elec Power (for melting) Savings = 14.0 − 8.0 (see p 10) (Max power now 6 MW)
= 6.0 MWh / cycle
& @ 2041 cycles / yr, Savings = 6 × 2041 = 12,246 MWh/yr. *

for 400 series, 22.2 NT FeCr Alloy + (48.4 − 13.6) NT Scrap
The 34.8 NT scrap is to be melted in 1 hour.
@ 450 Kwh/ton scrap ⇒ 34.8 × 0.45 = 15.7 MWh to melt
∴ Elec. Power (for melting) Savings = 21.8 − 15.7 (see p. 8) (Max power now 16 MW)
= 6.1 MWh / cycle.
& @ 1020 cycles/yr, Savings = 6.1 × 1020 = 6,222 MWh/yr. *

Induction Furnace Power Savings total = 18,486 MWh/yr.
see p.21 : 81,626 − 18,486 = 63,158 MWh/yr for Induction Furnaces.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 27 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

Revised Electric Power Usage: (see p.21)

Smelting Furnace	271,860	
Induction Furnaces	63,158	
Sub total	335,018	MWh/yr
+20% for Auxiliaries	67,004	
Total	402,022	MWh/yr

FeSi75 — Assume 50% savings, remainder used for alloying. (see p.21)

Annual FeSi75 Requirement = 0.50 × 4680
= 2340 NTPA.

Freight to Pittsburgh.

Class Rate for Slabs: $65.35/tonne
or ~ $60.00/NT

N.B. substantial reduction to be expected after a commodity rate has been secured.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 28 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

JANUARY, 1987 DATA. ($US).

Consumer Buying Prices for Scrap: $380 - 500/NT for 18-8
$90 - 150/NT for 410

List prices, FOB Mill. : $127/cwt for 304 Bar.
$62/cwt for 430 Sheet.
$77/cwt for 304 Strip

Typical US producer prices :-

$350 - 400/ NT 409 Slabs.
$750 - 850/ NT 304 Slabs

Therefore averaged producer price for 75% 304 slabs and 25% 409 slabs is :-

$\{(350 \times 0.25) + (750 \times 0.75)\} - \{(400 \times 0.25) + (850 \times 0.75)\}$

$= 650 - 738.$

Say $650 - 750/NT slabs.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 29 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

TO DETERMINE ACTUAL COST OF 300 vs 400 SERIES STEELS

300 SERIES (150,000 NTPA ≡ 27,300 NTPA Cr.)		400 SERIES (50,000 NTPA ≡ 6,000 NTPA Cr.)
$ 9,909,295	Chromite Concentrates	$ 2,202,065
$ 9,973,636	{Bentonite, Coke Fines + Lump, Quartz, Limestone / Dolomite, Elec. Paste}	$ 2,216,364
$ 6,374,846	Electric Power	$ 2,469,638
$ 6,439,485	Steel Scrap	$ 4,960,515
$45,000,000	Nickel	—
$ 318,000	Oxygen	$ 107,850
$ 17,850	Nitrogen	$ 6,750
$ 1,032,500	Argon	$ 516,250
$ 4,000,500	Lime / Spar / Alloys	$ 1,333,500
$17,133,000	{Labor, Mtce Supplies, Misc. Supplies & Serv., Office / Overhead}	$ 5,711,000
$100,019,112		$19,523,932
$ 666.79 /NT	WEIGHTED AVG. COST $597.69/NT (CAN)	$ 390.48/NT
$ 60.00	Freight to Pittsburgh	$ 60.00
$746.79 /NT (CAN)	"Producer Cost" at Pittsburgh.	$450.48/NT (CAN)
or $560/NT (US).		or $338/NT (US)
$750 – $850/NT.	Typical US producer production costs.	$350 – 400/NT

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 30 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

TO DETERMINE STAND-ALONE FeCr PLANT.

CAPITAL: $100 Million (Canadian)

96,620 NTPA Molten FeCr ≅ 81,500 NTPA Saleable FeCr
(~90% Rec: molten → saleable)

Chromite Concentrates	$ 12,111,360
Bentonite, Coke Fines & Lump, Quartz, Dol/Limestone & Elec Paste.	$ 12,190,000
Electric Power	$ 6,716,644
Labor (52% of total)	$ 4,461,854
Mtce. Supplies, Mis. Supplies & Services. (41%)	$ 5,125,000
Office/Overhead.	$ 1,100,000
Freight ($45/NT × 81,500 NTPA)	$ 3,667,500
	$ 45,372,358

Production: $81{,}500 \times 0.401 \times 2000 = 65.36 \times 10^6$ pounds Cr /yr.

$\Rightarrow$ production cost, delivered to Pittsburgh $= \dfrac{45{,}372{,}358 \times 100}{65.36 \times 10^6}$

$= 69.4\ ¢(CAN)/lb.$

or $52\ ¢(US)/lb.\ Cr.$

Capital Charge $= \dfrac{100 \times 10^6 \times 0.20 \times 100}{65.36 \times 10^6} = 30.6\ ¢(CAN)/lb$

or $23¢(US)/lb\ Cr$

$\Rightarrow$ Total cost to produce FeCr only from Bird River concentrates
$= 52 + 23 = 75\ ¢(US)/pound\ Cr$

Current selling prices: 32–38 ¢(US)/pound Cr.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 31 OF 31

DESCRIPTION	PROJECT NO.	MADE BY / DATE	CHECKED BY / DATE

IMPACT OF THE COST OF CHROMITE CONCENTRATES ON THE TOTAL PRODUCTION COST (excl. Freight)

Basis: 30% Cr_2O_3, 0.84:1 Cr:Fe, $630 CIF (CAN)/NT

Concentrates	:	100%
FeCr (40% Cr) (see p. 30)	:	29%
400 series Stainless Slab. (12% Cr) (see p. 29)	:	11%
300 series Stainless Slab. (18% Cr) (see p. 29)	:	10%

APPENDIX II

HATCH ASSOCIATES CONSULTANTS, INC.
ESTIMATE DETAILS

ACC'T Nº
PAGE 1 OF 8
DATE 3/87.
BY DPJ's
PROJ.Nº 9075

PROJECT BIRD RIVER CHROMITE

SECTION SUMMARY

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
A	PELLETIZING AND SINTERING				$ (CANADIAN)		$10.
B	SMELTING						960.
C	SCRAP MELTING + HOLDING						$1[illegible].
D	AOD REFINING						$.2
E	SLAB CASTING						70 x 10
	SUB-TOTAL						170 x 10
	CONTINGENCY - 20%						35 x 10^6
	EPCM - 15%						30 x 10^6
	WORKING CAPITAL						15 x 10^6
	TOTAL CAPITAL COST						250 x 10^6

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

PAGE 2 OF 8

PROJECT BIRD RIVER CHROMITE.

DATE

SECTION A : PELLETIZING & SINTERING.

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
1	SLURRY DAYTANK (600 NT. SOLIDS IN SUSPENSION)						
2	RECIRC & DELIVERY PUMPING SYSTEM						
3	12 FT. x 16 FT. DRUM FILTER (26 NTPH CAKE DISCHARGE)						
4	CONVEYOR FOR CAKE TO MIXER						
5	100 NT CAPACITY STORAGE BIN FOR BENTONITE & 1000 lbs/hr ADDITION SYSTEM						
6	100 NT CAPACITY STORAGE BIN FOR COKE FINES & 1500 lbs/hr ADDITION SYSTEM.						
7	MIXER — 28 TPH CAPACITY						
8	CONVEYOR FOR MIXED MATERIALS TO PELLETIZIER.						
9	WATER ADDITION SYSTEM AT 10 USG.P.M.						
10	DISC PELLETIZER — 30 TPH GREEN PELLETS						
11.	CONVEYOR FOR GREEN PELLETS TO SINTERING FURNACE.						
12	8 FT x 16 FT x 45 FT HIGH SINTERING FURNACE, REFRACTORY LINED, BURNERS FOR FURNACE GAS WITH HOODS, COOLING AIR BLOWERS FEED: 30 NTPH; PRODUCT: 26 NTPH.						

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

PROJECT BIRD RIVER CHROMITE

DATE

SECTION A: PELLETIZING & SINTERING (CONT.)

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
13	CONVEYOR FOR COOLED FIRED PELLETS TO STORAGE AREA						
14	COVERED STORAGE FOR FIRED PELLETS						

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

PROJECT BIRD RIVER CHROMITE

DATE

SECTION B : SMELTING

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
1	COVERED STORAGE FOR FLUXES, REDUCTANTS AND RECYCLE MATERIALS						
2	RECLAIM CONVEYORS FOR RAW MATERIALS AND PELLETS TO STORAGE BINS						
3	BATCHING PLANT						
4.	CONVEYORS FOR RAW MAT'L BATCHES TO FURNACE BINS.						
5.	FURNACE FEED CHUTES						
6.	39 FT. O.D. FURNACE SHELL, C/W REFRACTORY LINING						
7.	CLOSED SMOKE-HOOD.						
8.	3 x 5.2 FT (1.6m) DIAM SODERBERG ELECTRODE COLUMNS.						
9.	3 x 16 MVA FURNACE TRANSFORMERS.						
10.	OFF-GAS DUCTS TO VENTURI SCRUBBER.						
11.	SCRUBBER WATER RECIRC. SYSTEM AND DUST REMOVAL						
12.	OFF-GAS BOOSTER FANS						
13.	GAS HOLDER.						
14.	DRILL & MUDGUN FOR TAPPING						

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

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PROJECT BIRD RIVER CHROMITE

DATE

SECTION B : SMELTING

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
15	4 x 40 TON LADLES						
16	HOT METAL CRANE: 80TON CAP.						
17	2 x LADLE HEATERS FIRED BY FLE GAS						

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

PROJECT BIRD RIVER CHROMITE

DATE

SECTION C: SCRAP MELTING & HOLDING.

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
1.	SCRAP STORAGE AND RECLAIM SYSTEM						
2.	SCRAP BINS						
3.	DISCHARGE CONVEYORS TO INDUCTION FURNACES						
4.	2 x 75 TON INDUCTION FURNACES.						
5	2 x ? MVA FURNACE TRANSFORMERS.						
6.	SMOKE-HOODS AND DUCTS TO BAGHOUSE.						
7.	3 x 80 TON TRANSFER LADLES						
8	HOT METAL CRANE: 120 TON CAP.						
9.	REBRICKING ARRANGEMENT FOR 40 TON & 80 TON LADLES						
10	LADLE PREHEATING FOR EATON LADLES.						

HATCH ASSOCIATES CONSULTANTS, INC.

ESTIMATE DETAILS

ACC'T Nº

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PROJECT BIRD RIVER CHROMITE

DATE

SECTION D: AOD REFINING.

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST MAT.	LAB.	TOTAL
1	ALLOY, FLUX AND SCRAP. STORAGE & ADDITION SYSTEM						
2.	75 TON AOD INSTALLATION, 2 VESSELS, 1 BASE, C/W MECHANICALS, TOP LANCE, ETC						
3.	ARGON, OXYGEN, NITROGEN GAS MIXING & DELIVERY SYSTEM.						
4.	OFF-GAS DUCT AND BAGHOUSE						
5.	VESSEL REBRICKING ARRANGEMENT						
6.	VESSEL PRE-HEATERS.						

HATCH ASSOCIATES CONSULTANTS, INC.
ESTIMATE DETAILS

ACC'T Nº

PROJECT BIRD RIVER CHROMITE

DATE

SECTION E: SLAB CASTING

BY

PROJ. Nº

ITEM Nº	DESCRIPTION	QUANT	UNIT	RATE	ESTIMATED COST			
						MAT.	LAB.	TOTAL
1.	120 TON CAP. CASTER CRANE							
2.	LADLE HOLDING SYSTEM							
3.	TUNDISH ARRANGEMENT							
4.	75 NTPH SLAB CASTER - COMPLETE (46 inch WIDE x 8-10 inch thick)							
5	CUT-OFF SYSTEM							
6	RUN-OUT TABLES							
7	SLAB HANDLING CRANE							
8	SLAB STORAGE AREA.							